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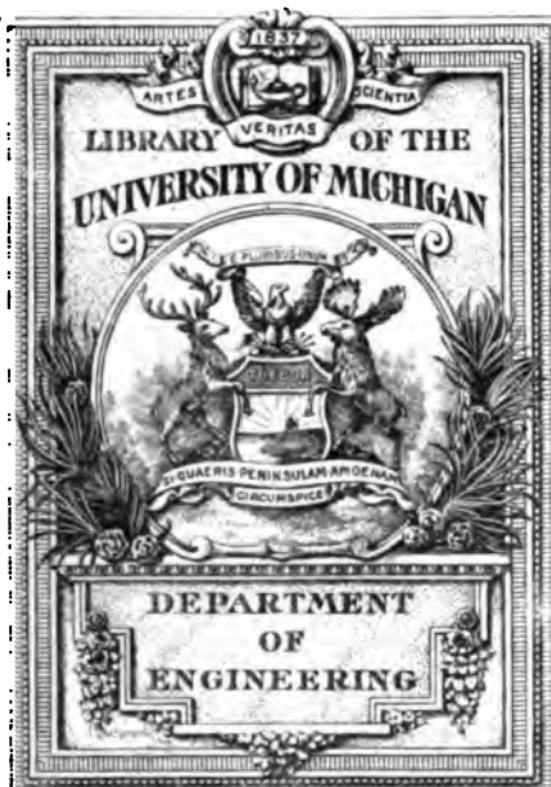
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**WORKS OF
A. W. FAHRENWALD**

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Its Control and Operation

viii + 256 pages, 4 by 6 $\frac{1}{2}$, 37 figures, flexible
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THE CYANIDE PROCESS ITS CONTROL AND OPERATION

BY

A. W. FAHRENWALD, E.M., Met.E.

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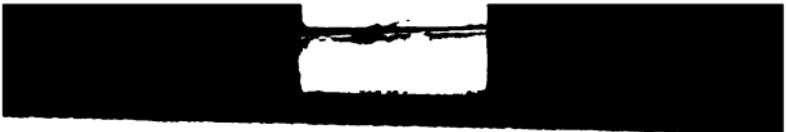
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**TO THE MEMORY OF
HIS MOTHER**

**THIS BOOK IS LOVINGLY DEDICATED BY
THE AUTHOR**

530313





PREFACE

WHILE the cyanide process has been somewhat curtailed by flotation methods, it still remains the most important process for the extraction of gold and silver from their ores. Its recent developments, particularly from the laboratory point of view, have brought out much new data, due partly to the abnormal conditions resulting from the world-wide war. These conditions have had little effect upon the mechanical side of cyanidation, but most of the chemicals used in the process have been directly affected, and some of them, after much investigation, have given way to less expensive ones.

Much has been written on the cyanide process, but no book now available attacks the subject from the point that is most vital, namely, the laboratory. Every successful process owes its existence to a laboratory, which works out the details of efficient treatment and alters them as conditions demand. The object of this book is to furnish a laboratory guide, both for investigating a new ore and for conducting the laboratory of an operating mill. It should be helpful to the student and beginner, and it is hoped that it may also be of assistance to the more experienced.

In the ordinary laboratory one does not hav-

resource to the widely scattered material bearing on testing operations; generally, small mill laboratories are remote from such sources of information. The effort has been to make the book thorough from this standpoint, and to arrange the material in a logical manner.

I wish to acknowledge my indebtedness to others who have been active in this field of metallurgy, to whose works frequent reference has been made and from which much valuable material has been taken.

A. W. FAHRENWALD.

EL PASO, TEXAS,
March, 1918.

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INTRODUCTION

It has often been stated that the cyanide process would be revolutionized by the flotation process. This does not seem to have been the case. The two processes have separate and distinct objects in view, and it is probable that the cyanide process will remain the leading method for extracting gold and silver from their ores. Flotation may replace the cyanide process in some particular cases; for example, with an ore in which the precious metal is held in combination with an easily floatable mineral, the gold and silver can be obtained in a much smaller bulk of material than before concentration, after which they may be smelted or treated by the cyanide process, as has been attempted, in some cases, with satisfactory results. Slime concentrate may be removed by flotation, which formerly could not be completely recovered. Under some conditions metallic gold and silver are thus obtained. The ordinary run of siliceous ores will no doubt continue to be treated by the cyanide process.

The cyanide process, as we know it, owes its existence to J. S. MacArthur, R. W. Forrest, and W. Forrest. The solvent action of cyanide upon gold and silver had been known to other sci-

tists and metallurgists, but as they had no direct influence upon the present process they will not be discussed. The MacArthur-Forrest process is best described by John S. MacArthur.¹ Their invention was covered by two British patents, No. 14,174 of October 19, 1887, and No. 10,223 of July 14, 1888. The principal points covered in these patents are:

1. Treatment of the powdered ore with a weak cyanide solution.
2. Drawing off the gold-bearing solution from the treated ore.
3. Precipitation of the gold by zinc.

These three operations form the basis of the present cyanide process. The mechanical details of the process were rapidly improved, and the underlying chemistry was fairly well ascertained, considering the numerous and complex reactions that took place during the early years of the process. In later years, investigation of the chemical side has been comparatively slow, although this is becoming quite thoroughly understood.

¹ *Min. & Sci. Press*, June 10, 1916.

CYANIDE PROCESS

CHAPTER I.

PROPERTIES OF GOLD AND SILVER.

GOLD.

Physical Properties.—Gold is the only metal that has a yellow color when massive and pure. The color is greatly modified by impurities, silver lowering the tint, while copper heightens it. When finely divided it may assume violet, ruby, purple, dark brown, or black colors.

Malleability and Ductility.—Gold exceeds all other metals in malleability and ductility. Leaves not more than $1/300,000$ of an inch in thickness can be obtained by beating, and a single grain can be drawn into a wire 500 feet long. The presence of 0.05 per cent of bismuth makes gold so brittle that it can almost be crumpled in the fingers. Lead and tellurium have a similar effect.

Hardness.—Gold is softer than silver, but harder than tin.

Tenacity.—Pure gold has a tenacity of 7 tons per square inch, with an elongation of 30.8 per cent.

2 PROPERTIES OF GOLD AND SILVER

Specific Gravity. — Cast gold varies from 19.29 to 19.37.

Fusion Temperature. — The melting point of gold is approximately 1063° C., just below that of copper, and about 100° higher than that of silver.

Volatilization of Gold. — Pure gold, according to Rose, begins to volatilize in ordinary furnaces at a temperature just below 1100° C.; the loss per minute at 1250° C. is four times as great as at 1100° C. In ordinary melting furnaces the total amount of loss probably seldom exceeds 0.1 per thousand in melting a charge of 1200 oz. The rate of loss during melting does not depend entirely on the temperature. A current of air passing over the surface of the molten metal increases the amount of volatilization, so that if the lid of the crucible is removed for any purpose the loss is increased.

The presence of volatile impurities in bullion also causes increased loss. Lead only very slightly increases the volatility of gold. Copper and zinc have a more marked effect, while 5 per cent of antimony or mercury will cause a loss of 2 parts per thousand at 1245° C. According to Rose's experiments at the Royal Mint, tellurium does not cause volatilization of gold below 1100° C. Hence the losses incurred in roasting telluride ores are not due to volatilization.

Electric Conductivity. — The conductivity of gold is 76.7, that of silver being 100.

Atomic Weight of gold is 197.2, oxygen being 16.

Effect of Foreign Substances. — Even when present in minute quantities, Pb, Bi, Cd, Sb, As, and Sn produce brittleness; tellurium has the same effect. Osmium and iridium as osmiridium do not alloy with gold, but remain as hard grains in the mass, and are found chiefly at the bottom of ingots.

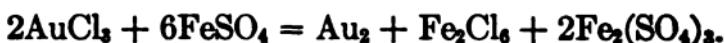
Chemical Properties. Solvents. — The chief solvents of gold in industrial use are "aqua regia," solutions of chlorine, and potassium cyanide. It is also soluble in any mixture producing nascent chlorine, such as solutions containing mixtures of (1) nitrates, chlorides, and acid sulphates; (2) chlorides and some sulphates, for example, ferric sulphate; (3) hydrochloric acid and nitrates, peroxides, such as permanganates, or chlorates; (4) bleaching powder and acids.¹ Gold is soluble in solutions of bromine, or of mixtures producing bromine, and in boiling concentrated solutions of ferric chloride. Iodine dissolves gold only if it is nascent, or if dissolved in an iodide, or in ether or alcohol. Gold is also soluble in solutions of the thiosulphates (hyposulphites) of calcium, sodium, potassium, and magnesium, in the presence of an oxidizing agent. Sulphuric acid, hydrochloric acid, and nitric acid separately have but little action on gold. It is not perceptibly attacked by alkalies. If nitric and sulphuric acids are present together, gold is dissolved. It is also attacked by alkaline sulphides, producing gold sulphide, which is dissolved.

¹ Rose, "Gold," pp. 9-12.

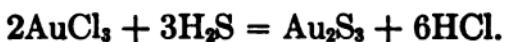
4 PROPERTIES OF GOLD AND SILVER

Precipitants.—The chief precipitants of gold in industrial use for obtaining the metal from solutions are.

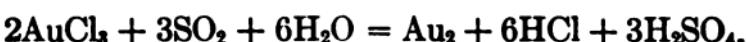
1. Ferrous sulphate, used on a large scale for precipitating from solutions of gold chloride at chlorination mills, and in the "aqua regia" parting process. The reaction is:



2. Sulphuretted hydrogen, used also at chlorination mills, and in the von Patera process for silver ores when gold is present. The following equation represents the reaction:



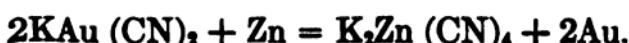
3. Sulphurous acid (sulphur dioxide), in use chiefly in laboratories for the preparation of pure gold from chloride solutions, but not on a large scale, as precipitation is slow until the solution is saturated with the gas.



Oxalic acid is sometimes used in the laboratory for the same purpose.

4. Charcoal, formerly used, on the largest scale, at the chlorination works at Mt. Morgan (Queensland). The reaction has not been fully investigated. It has also been used at some small cyanide plants.

5. Zinc is extensively used in the form of extremely thin shavings or dust in cyanide plants for precipitating the gold from cyanide solutions, the reaction being:



Gold is also separated from cyanide solutions by electro-deposition.

Gold Chlorides. — There are three chlorides, AuCl , Au_2Cl_4 , and AuCl_3 . Only the latter, auric chloride, or gold trichloride, is of metallurgical importance. Gold trichloride is very soluble in water. In the air it is readily decomposed by heat, the decomposition being complete at 220° to 230° C. with the formation of gold and chlorine. Even at 100° C. it is decomposed by light, whether dry or in solution, but not when free hydrochloric acid is present.

The volatilization of gold trichloride, which is of importance in the roasting of ores with salt, has been found by Rose to begin at 180° C. and to reach a maximum at about 290° C. It then falls off gradually up to 800° C., when it again increases. Between 500° and 800° (very dull red to red heat) the volatilization is small.

Solutions of gold trichloride are decomposed and the gold precipitated by many reducing agents, as, for example, sulphur dioxide, oxalic acid, ferrous sulphate, ferrous chloride, phosphorus, and most organic substances and metals, the decomposition in every case being hastened by heating. They are reduced very rapidly by zinc and iron, and more slowly by lead, gold being precipitated. Sulphuretted hydrogen completely precipitates the gold as sulphide, from both acid and neutral solutions.

Detection of Gold as Chloride. — Ferrous sulphate is frequently used to detect the presence of gold in solution as chloride. Two test t

6 PROPERTIES OF GOLD AND SILVER

or Nessler tubes are placed side by side on a sheet of white paper. One is nearly filled with the solution to be tested, the other with water, and a few drops of a solution of ferrous sulphate are added to each. On looking down the tubes a brown coloration, due to finely divided gold in suspension will be seen in the tube containing the chloride solution. In this way the presence of only 1/720,000 part of gold in the solution (1 dwt. per ton of water), or even less, can be detected.

A still more delicate test is afforded by the reaction of a mixture of SnCl_2 and SnCl_4 in solution with AuCl_3 , by which a precipitate of purple of Cassius is produced. The solution containing AuCl_3 is heated to boiling and poured suddenly into a large beaker containing 5 to 10 c.c. of a saturated solution of the stannous chloride, when a precipitate is at once produced which will be tinted purplish-red or blackish-purple, even if the solution contains as little as 1 part of gold in 5,000,000 parts of water (3.5 grains per ton). By this test Dr. Rose has detected the presence of 1 part of gold in 100,000,000 parts of water (1 grain in 6 tons).

Hyposulphites of Gold. — These are of importance to the metallurgist in connection with the extraction of gold from silver ores by leaching with sodium or calcium hyposulphites, and by the Russell process.

Alloys of Gold. — Although gold alloys with most metals, only a few of the alloys are of practical use in the arts or manufactures. The most

important are those which gold forms with mercury, with copper, and with silver.

Amalgams. — Gold is soluble in mercury, the alloys being termed amalgams. According to Kasentseff,¹ mercury dissolves 0.11 per cent of gold at 0°; 0.126 per cent at 20°; and 0.65 per cent at 100°. Gold, however, unites with mercury to form a white solid amalgam containing about 13.5 per cent of gold. When excess of mercury is present, as in the amalgamation of gold ores in mills, this amalgam will be suspended as particles in the mercury, and can be separated by straining it through chamois leather. The amalgam thus obtained will always contain some mercury, and will, moreover, be very variable in composition, as it always contains particles of gold which have been partly penetrated by mercury. The gold will range from 25 to 50 per cent, being highest when the ores contain the metal in coarse grains. When the amalgam is heated by distillation in retorts, as at mills, the mercury is volatilized, condensed, and collected, but at least 0.1 per cent is left in the gold, and is expelled only when the gold is melted.

Gold and Silver. — Gold and silver unite in all proportions, forming homogeneous alloys which, as the proportion of silver is increased, become lighter in color until, when 60 per cent is present, they are quite white. They are all malleable and ductile.

When the alloys contain not less than 60 per cent of silver, the silver may be almost completely

¹ *Bull. Soc. Chem.*, Vol. XXV, p. 20.

8 PROPERTIES OF GOLD AND SILVER

dissolved out by nitric or by sulphuric acid; but if less is present the silver is only partly removed. Aqua regia attacks these alloys, but its action is soon stopped by the coating of AgCl which forms on the surface of the metal.

The alloys are used in jewelry, as, for example, "green gold," containing 25 per cent of silver; but triple alloys of gold, silver, and copper are most common. Alloys of gold with various proportions of silver, from 35 to 86 per cent, have been largely used in Japan for the old gold coinages and for gold jewelry, a pure gold surface being obtained by dissolving out the silver from the upper layers of the alloy by pickling processes.

Gold and Copper. — Gold alloys with copper in all proportions, giving alloys which are harder, less malleable and ductile, but more elastic than the unalloyed metal. When the metals are of ordinary purity, segregation on solidification takes place to such a slight extent that it may be disregarded. The presence of lead or bismuth, however, will cause segregation. Gold-copper alloys containing excess of copper are acted on by nitric and sulphuric acids in a similar manner to gold-silver alloys, the copper being dissolved.

The gold-copper alloys are in extensive use for coinage and jewelry. In England, standard gold contains 916.6 parts of gold and 83.4 parts of copper, and is that adopted for the gold coinage. The gold coins of Russia, Portugal, Turkey, and India are also of this composition. In France, America, and most other countries, the standard is 900 parts of gold and 100 parts of copper. The

Egyptian standard has a fineness of 875, while the Austrian ducat is 986, and that of Holland 983. The English and French standards are the best for coinage, the latter resisting wear more than the former.

For gold wares in England the standards recognized by law are those containing 22, 18, 15, 12, and 9 parts of gold in 24 parts. These are designated by the term "carat," an alloy of 22 parts of gold and 2 parts of copper being called 22 carat gold. The best jewelry is of 18 carat, but 22 carat is used for wedding rings.

Gold alloys with iron, and the more readily if the molten gold is at a high temperature; hence the metal should not be stirred, when molten, with an iron rod.

Segregation in Gold Alloys. — The alloys in which segregation is of special importance to metallurgists are those of gold, silver, lead, zinc produced in cyanide mills. This has been investigated by Edward Matthey,¹ with the following results: One such ingot, which weighed about 120 oz., gave 662 parts of gold per thousand at one of the bottom corners, and only 439 parts at the top. Another, which weighed 400 oz., and contained 16.4 per cent of lead and 9.5 per cent of zinc, yielded 614.0 parts of gold per thousand and 75.8 parts of silver by actually separating the whole of the precious metals, its true value being £1028. The value deduced from the average of 14 assays made on it (gold 576.0, silver 90.0) was only £965. Seven dip assays made on this ingot

¹ Proc. Roy. Soc., Vol. LX (1896), p. 21.

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varied from 562 to 622 parts of gold per thousand. The segregation in other ingots was even more remarkable.

In experiments on alloys of gold and zinc, Matthey found that gold tends to segregate toward the centre of the masses, but only in a slight degree; the centre of a sphere, 3 inches in diameter, of an alloy containing gold 90, zinc 10, was only 1 to 2.5 parts per thousand richer than the outside. In alloys of gold and lead the segregation was much greater, the centre of a 3-inch sphere being 29 parts per thousand richer than the outside when 30 per cent of lead was present.

A still greater segregation takes place when lead and zinc are present together; thus in an alloy containing 15 per cent of lead and 15 per cent of zinc, the sphere contained 657 parts of gold at the top, 785 in the centre, and 790 at the bottom, gravity thus producing an effect. If, however, silver is present to the amount of two-thirds of the quantity of zinc and lead together, segregation does not take place. An alloy containing approximately gold 55, zinc 7, lead 18, silver 20 was found to be practically homogeneous.

Matthey¹ also investigated the segregation which occurs in gold ingots containing members of the platinum group. When such gold is cooled from a state of fusion, an alloy rich in the more fusible element (gold) falls out first, driving the less

¹ *Proc. Roy. Soc.*, Vol. LI (1892), p. 447; and *Phil. Trans. Roy. Soc.*, Vol. CLXXXIII (1892), A, pp. 629-
630.

fusible constituent to the centre. Thus the assay of an outside cut of such an ingot gives a result too high in gold, sometimes by several per cent.

SILVER.

Physical Properties. — Silver is the whitest of metals, and is capable of receiving a bright polish. It is harder than gold, but softer than copper.

Malleability and Ductility. — With the exception of gold, silver exceeds all other metals in these properties. It is said that it can be hammered into leaves only 1/1000 of an inch thick, and a single grain can be drawn into 400 feet of wire.

Tenacity. — The tenacity of silver at 0° C. is about 18 tons per square inch (Baudrimont).

Specific Gravity. — The specific gravity of cast silver is 10.50, and after being struck in a coining press, 10.57.

Effect of Heat. — The melting point of silver is 962° C., and its boiling point 1850° C. (Moissan).

When silver is melted in crucibles there is a slight volatilization if it is heated much above its melting point. Its volatility is much increased in the presence of Pb, Zn, As, and Sb, and to this the loss of silver in roasting furnaces, in which the temperature is never sufficiently high to volatilize the pure metal, is largely due.

When silver is melted in the air, it absorbs about 22 times its volume of oxygen, which is then given off with effervescence as the metal solidifies. When cast in an ingot mould, the upper surface of the ingot will be covered with protuberances

12 PROPERTIES OF GOLD AND SILVER

resembling miniature volcanoes, and particles of silver will be projected to some distance. This phenomenon is termed "spitting" or "sprouting"; it occurs only when the silver is comparatively pure, and is prevented by the presence of small quantities of copper or lead, or by stirring the metal under a cover of charcoal. "It is probable that the oxygen exists in the molten silver in the form of the oxide, Ag_2O , as this substance has been shown to be stable at very high temperatures, although it is instantly decomposed at a red heat" (Rose).

Electrical Conductivity. — The conductivity of silver for electricity is higher than that of any other metal.

Atomic Weight. — The atomic weight of silver is 107.88.

Chemical Properties. *Solvents.* — Silver is not acted on, at ordinary temperatures, either by dry or moist air. The blackening of silver articles in the air of towns is due to the formation of a film of silver sulphide from the presence of traces of H_2S .

Alkalies or alkaline carbonates have no action on silver, even when molten. Nitric acid dissolves it readily, and is the best solvent. It is soluble in boiling concentrated sulphuric acid, but not in the dilute acid. Hydrochloric acid acts on it very slowly and only superficially, forming an impervious coating of chloride. It is not attacked by other acids, but chlorine, bromine, and iodine act on it with the formation of AgCl , AgBr , and AgI . It is also soluble in cyanide

solutions when in fine particles. Silver also dissolves in solutions of sodium hyposulphite as a double hyposulphite of silver and sodium. When finely divided silver is heated with common salt, silver chloride is formed.

Silver Oxides. — There are two oxides of silver, AgO and Ag_2O , but they are of little metallurgical importance.

Silver Sulphide, Ag_2S . — On the properties of the sulphide and chloride the most important processes for the extraction of silver depend. The sulphide is dark gray to black, and is found native as the mineral argentite. Silver has a strong affinity for sulphur, and readily combines with it.

Silver sulphide is practically non-volatile at high temperatures, and remains unaltered if air be excluded, but in the presence of air it is decomposed into metallic silver and sulphur dioxide. It is insoluble in solutions of the alkalies, their carbonates, and chlorides, and in sodium thiosulphate (hyposulphite), but is soluble in solutions of potassium and sodium cyanides, and of double hyposulphite of sodium and potassium. It is also soluble in nitric acid, sulphur being separated and silver nitrate formed, and is decomposed by strong sulphuric and hydrochloric acids.

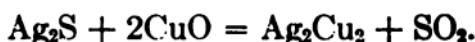
Sulphide of silver is soluble in molten copper sulphide, lead sulphide, and other metallic sulphides, forming argentiferous mattes, and on this property the extraction of the metal by matte smelting processes from copper and other ores depends. Metallic silver, silver chloride, and bromide also dissolve in mattes as sulphide.

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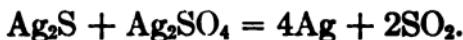
In amalgamation processes, mercury decomposes the sulphide, the resulting silver forming an amalgam with the excess of mercury present. The reduction of the silver is greatly facilitated when a solution of alum, ferrous sulphate, or cupric sulphate is added.

Sulphide of silver is completely reduced by iron at a red heat, but the iron must be in slight excess (Karsten); copper and lead effect only partial reduction. When heated with an excess of nitre, the whole of the silver is reduced, but with alkaline carbonates and cyanides the reduction is not complete. When heated in air, it is reduced to metallic silver, no sulphate being formed.

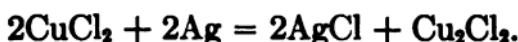
If silver sulphide be fused with PbO or CuO , decomposition takes place as follows:



If it be fused with Ag_2SO_4 , metallic silver is produced:



Silver Chloride, AgCl . — This occurs native as the mineral kerargyrite. Silver chloride is largely produced in many metallurgical operations, such as chloridizing, roasting, etc. It is also formed by the action of cupric, ferric, and mercuric chlorides on silver, thus:



It is readily soluble in hot solutions of sodium thiosulphate, ammonium, calcium, and magnesium chlorides, alkaline cyanides, and hot con-

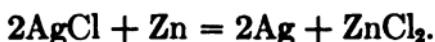
centrated hydrochloric acid; it is almost insoluble in water.

Silver chloride is completely reduced when fused with alkaline hydrates and carbonates, sodium carbonate being generally employed, thus:

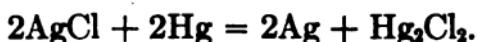


When fused with 1 part of charcoal to 2 parts of chloride, the silver is reduced.

In the wet way zinc and iron are the best reducing agents:



Copper reduces it perfectly from ammoniacal solutions, and from strong sodium chloride solutions, but not satisfactorily from acid solutions. Mercury reduces it slowly, according to the equation:



The action of mercury is promoted by the presence of cupric sulphate, ferrous sulphate, and alum; also by the presence of iron.

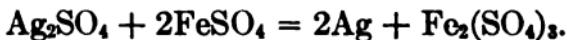
Silver Sulphate, Ag₂SO₄. — This is extensively produced in the operation of "parting." It is soluble in hot water, but the best solvent is sulphuric acid of specific gravity 1.25; it is only slightly soluble in cold water. The silver is completely precipitated from solutions of silver sulphate by Cu, Fe, Zn, Sn, and Pb. In solutions obtained in gold and silver parting by sulphuric acid, and in those of the Ziervogel process, copper is always used as the precipitant.

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Silver sulphate is completely reduced by charcoal at a dull red heat, as in the new Gutzkow process, the reaction being:



The crystals are also reduced in the old Gutzkow process by ferrous sulphate:



Silver sulphate is decomposed at a bright red heat into silver, oxygen, and sulphur dioxide.

Silver Bromide, AgBr . — This occurs native as the mineral bromite. It is soluble in most of the solvents of silver chloride, but generally to a less extent. At the Utrecht Mint, sodium bromide is used instead of sodium chloride for the precipitation of silver in the volumetric process of assay.

Silver Iodide, AgI . — This occurs native in the mineral iodite. It is soluble in solutions of alkaline cyanides and hyposulphites, but only very slightly in solutions of alkaline chlorides.

Silver Cyanide. — The double cyanides of silver and potassium, $\text{KAg}(\text{CN})_2$, and of silver and sodium, are of great importance in the metallurgy of silver, since the introduction of the cyanide process for the extraction of silver from its ores. Silver is precipitated from their solutions by zinc, copper, and aluminium, and by electricity.

Silver Bullion. — Silver bullion is the term applied to silver in the mass — *i.e.*, in ingots or — as distinct from coin or plate. When it contains gold it is called “doré” silver, and then is bought by gold and silver refiners. “Refined”

silver is silver nearly pure and practically free from gold. It usually contains 997.5 to 998.5, or occasionally 990.0 parts of silver in 1000 parts, when it is said to be 997.5, 998.5, or 990.0 fine. "Fine" silver is chemically pure metal.

In the London market the price is quoted per ounce Troy, in terms of standard silver (925 silver per 1000). In New York the price quoted is for fine silver. The impurities generally present are copper, lead, and sometimes zinc and bismuth.

Industrial Alloys of Silver. *Silver and Copper.*—Silver can be alloyed with copper in all proportions, but the alloys are not homogeneous except the eutectic. The eutectic alloy (71.89 per cent of silver), which has been termed Levol's alloy, is perfectly homogeneous whether slowly or rapidly cooled. It has been used for coinage in the Dutch Indies. Ingots containing more or less silver than the eutectic are not of uniform composition, the former being richer at the centre than at the sides, and the latter richer at the sides than at the centre. This fact is of great importance in the valuation of silver bullion, as it determines the manner in which the samples for assay should be taken. In such bullion it is necessary to prepare the assay piece by dipping out a small quantity of the molten metal with a charcoal dipper and granulating it by pouring it into water, not by taking a cutting from the ingot. The granules should be well dried on a hot plate, after they have been hammered flat, as they sometimes, but rarely, are hollow spheres and contain water. "Dips" are rarely necessary in

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silver bullion when the fineness exceeds 975, and assay pieces can be cut from one of the bottom edges of an ingot, preferably at a point one-third distant from one of the ends.

Silver-copper alloys are in extensive use for coinage and silver wares of all kinds.

In England, the alloy for coins, plate,¹ and other articles, consists of 925 parts of silver and 75 parts of copper in 1000, and is known as "standard silver." In the United States, France, Germany, Italy, Japan, and other countries the alloy for the principal coins contains 900 parts of silver and 100 parts of copper, and is slightly harder than the English standard alloy. Alloys containing less silver are in use for the subsidiary coins in many countries except England; in France, Italy, Spain, Switzerland, etc., an alloy containing 835 parts of silver is employed, while in Sweden, Norway, and Denmark some denominations of the coins have only 400 parts of silver in 1000 parts.

Silver and Zinc. — Alloys of silver and zinc have been investigated by Peligot with a view to their employment for coinage, as they do not tarnish. He found that those containing from 13 to 20 per cent of zinc were fairly malleable, and an alloy of 80 per cent of silver and 20 per cent of zinc was not discolored even by alkaline sulphides. Zinc has a strong affinity for silver, and will remove silver from lead at a temperature a little

¹ There is also another alloy for plate, containing 958.3 parts of silver, but it is seldom used, as it is less durable than "standard" silver.

above its melting point; on this property the Parkes process for the desilverization of argentiferous lead depends.

Silver and Cadmium. — These alloys are softer, whiter, and more malleable and ductile than silver-copper alloys. The alloy consisting of 925 parts of silver and 75 parts of cadmium has been shown by Dr. Kirke Rose to be perfectly homogeneous, and it has been proposed by him to use it for standard trial plates at the Royal Mint.

Silver and Lead. — Silver and lead, when melted together, unite in all proportions, but during solidification they separate. Hence argentiferous lead when cast into pigs is not of uniform composition throughout. Satisfactory samples for assay cannot, therefore, be cut from the pigs, and it is necessary to remelt them and take dip samples (Claudet).

When silver-lead alloys are melted at a red heat, with free access of air, the lead is rapidly oxidized, the silver remaining unaltered. The process of cupellation is based on these properties.

Lead has a stronger affinity for silver than copper, so that when an alloy of copper and silver, with excess of lead, is heated to a temperature between the melting points of copper and lead, the lead will liquefy, carrying the silver with it. The old liquation process, now abandoned, except in Japan, was based on this property.

Silver and Bismuth. — Silver alloys with bismuth in all proportions, and the latter metal may be separated by cupellation, but small amounts always remain in the silver, and produce brittleness.

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ness. Ingots of silver containing bismuth are not homogeneous, the portions which remain longest fluid being richer in silver than the others, such ingots, even when of high fineness, must therefore be assayed from dips, and not from cuttings.¹

Silver and Mercury. — Mercury dissolves silver, forming amalgams which are liquid or solid according to the percentage of silver they contain. The amalgam first formed on silver in contact with mercury does not readily dissolve in the surrounding mercury, but rather tends to hinder its further action; hence it is that, in the amalgamation of silver ores, grinding is so necessary to remove this layer and lay bare fresh surfaces of silver to the action of the mercury. It also follows that a siliceous gangue in ores facilitates and a clayey gangue retards amalgamation.

There seems to be only one definite amalgam, as when any liquid amalgam is squeezed through canvas or chamois leather the product remaining will always consist approximately of 30.4 per cent of silver and 69.6 per cent of mercury, which agrees with the formula Ag_2Hg_2 , with 4.6 per cent of free mercury.

¹ Gowland, Journ. Chem. Soc., Vol. LI (1887), p. 410.



CHAPTER II.

THE NATURE AND PROPERTIES OF CYANIDE.

Cyanide and Cyanogen. — Cyanogen is the compound radical CN, composed of one atom of carbon and one atom of nitrogen; when united with some metallic substance, as potassium or sodium, it makes cyanide, KCN or NaCN. When combined with hydrogen it forms hydrocyanic acid, HCN, often called prussic acid. The radical CN remains as such in all phases of the cyanide process, for in its chemistry this chemical union is never broken. It is apparent that the solvent action is due to this compound, for neither carbon nor nitrogen has any effect on gold or silver. The two elements are not easily disassociated.

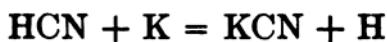
Properties. — Cyanogen (from *κυανός*, blue) owes its name to the fact that several of its compounds have a blue color. It is a colorless gas, easily soluble in water and alcohol, and extremely poisonous. It burns with a purple colored flame. In aqueous solutions, cyanide soon undergoes change and a brown amorphous substance is deposited. In the solution are found hydrocyanic acid, oxalic acid, ammonia, and carbon dioxide. The principal cause for this decomposition is apparently the tendency of the nitrogen to combine with hydrogen to form

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stable ammonia, and of carbon to combine with hydrogen and oxygen to form stable compounds like oxalic acid and carbon dioxide.

Cyanogen is a most active radical, especially toward metals, with which it forms several hundred compounds, thus increasing the difficulty in isolating and determining the properties of each, especially under working conditions. The cyanogen radical is related to the cyanides as chlorine is related to chlorides, or as the acid radical SO_4 is to sulphuric acid, H_2SO_4 . It unites with metals to form salts in the same manner as SO_4 and Cl unite with iron and sodium respectively, forming FeSO_4 and NaCl . The chemical principle involved is identical.

Cyanogen combines to form simple or single cyanides, which may be regarded as metals replacing the H of HCN , as:



and to form double cyanides, which may be considered as a combination of two single cyanides, as:

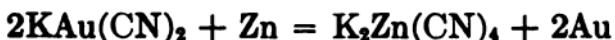


Many other complex compounds of cyanogen exist, and are found under working conditions.

In a salt of cyanogen, the metal can and will be easily replaced by one for which cyanogen has greater affinity. Thus, in KCN the potassium may be replaced by gold, forming AuCN and finally the double cyanide $\text{KAu}(\text{CN})_2$. If the solution containing gold cyanide be brought in contact with zinc, the cyanogen radical, having

POTASSIUM VS. SODIUM CYANIDE 23

a greater affinity for zinc than for gold, will form a double potassium zinc cyanide, as:



throwing down or precipitating gold. It is these reactions that make the cyanide process possible.

Source of Cyanogen in the Cyanide Process.

— Many cyanides of alkaline earths exist. A few are: sodium cyanide, NaCN ; potassium cyanide, KCN ; ammonium cyanide, NH_2CN ; barium cyanide, $\text{Ba}(\text{CN})_2$; calcium cyanide, $\text{Ca}(\text{CN})_2$; magnesium cyanide, $\text{Mg}(\text{CN})_2$; and strontium cyanide, $\text{Sr}(\text{CN})_2$. The sources of cyanogen in the cyanide process are sodium and potassium cyanides; all the others above mentioned have solvent powers similar to sodium and potassium cyanide but have not found application, mainly for economic reasons. Formerly, considerably more potassium cyanide than sodium cyanide was used, but in later years the reverse is true.

Potassium vs. Sodium Cyanide. — The only difference between these two is in their dissolving power. As this depends upon the cyanogen present, it will naturally be higher in sodium cyanide, the molecular weight of which is 49, as compared with 65.1 for potassium cyanide. The relationship may be shown thus:

$$\begin{array}{lll} \text{Na} & \text{C} & \text{N} \\ 23 + 12 + 14 = 49 & & 39.1 + 12 + 14 = 65.1 \\ \text{CN} = 53.06\% & & \text{CN} = 39.94\% \end{array}$$

Cyanogen thus constituting 53.06 per cent of sodium cyanide, as against 39.94 per cent ~~the~~

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potassium cyanide, gives NaCN a dissolving power 1.3285 times greater than that of KCN. Formerly the pure potassium salt was considered as 100 per cent strength, which would give NaCN a 132.65 per cent strength. This system is going out of use.

The Roessler & Hasslacher Chemical Company, New York, has issued the following circular, entitled "Cyanide."

Beginning January 1, 1916, we will change the designation of our various grades of cyanide, basing same on sodium cyanide content. When the cyanide now on the market superseded potassium, the various grades of sodium cyanide were designated by their equivalent in potassium cyanide, in order to show their comparative strengths. Potassium cyanide 95-96 per cent contains 38 per cent cyanogen. Thus sodium cyanide containing 51-52 per cent cyanogen was designated as 129 per cent strength, that being its equivalent in KCN. Owing to its lower cost, sodium cyanide has now entirely replaced potassium in recovery of precious metals, in fumigation, plating, etc., the results being equal if not superior to those obtained by the use of potassium cyanide. The lower cost of transportation is also an element of saving because of its greater cyanogen content. As KCN is no longer used, and the change to NaCN fully understood, a re-naming of the different grades of cyanide is advisable. We shall label and sell sodium cyanides as such.

CHAPTER III.

SUITABILITY OF AN ORE FOR THE CYANIDE PROCESS.

All gold and silver ores, if subjected to the cyanide treatment, will part with more or less of their gold and silver contents; in some cases a high extraction is obtained while in others only a few per cent of the values will be dissolved. The latter are known as refractory ores, and are characterized by some peculiar physical-chemical nature whereby the precious metals cannot be economically dissolved, generally because they are chemically combined or mechanically occluded or alloyed with some substance which prevents the dissolving action of cyanide. The ore, furthermore, may contain constituents which combine with cyanide in large quantities, or may cause precipitation of the precious metals from solution; these undesirable constituents are spoken of as "interfering elements." The result of an analysis, while it will throw light on the subject, will not indicate whether an ore can be cyanided or not. The only way of determining this is by laboratory tests. Refractory ores may be rendered suitable for the cyanide process by special treatment, or by modification of the usual method.

24 NATURE AND PROPERTIES OF CYANIDE

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Part 1

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Classification of Ores. — Ores may be divided into three main classes, as follows:

1. Clean siliceous ores, having no sulphides or base metals nor their compounds.
2. Base ores, containing unaltered sulphides or base-metal compounds.
3. Oxidized ores, resulting from oxidation of sulphides.

Ores of Class 1 seldom cause trouble; a high extraction is possible with little consumption of cyanide and no fouling of solutions.

Ores of Class 2 cyanide less easily, partly because the ore tends to alter and decompose, thus consuming cyanide and oxygen and introducing substances which may foul the solution and obstruct further dissolution of gold and silver and its precipitation. These ores may belong to the refractory class, and ordinarily require special methods.

With ores of Class 3, a good extraction is usually possible, owing to the fact that a large part of the troublesome constituents have been removed; the ore also is of a more porous nature and the precious metal is more easily attacked. Where the precious metal is associated with chlorides, carbonates, and partially oxidized minerals, representing an intermediate stage in the transition of sulphides to fully oxidized minerals, such ores are hard to treat, owing to the activity with which cyanide attacks these soft and easily decomposed base metals. Barite ores carrying silver are hard to treat.

Interfering Elements. Iron. — The effect of iron upon cyanide solution depends largely on

the state of its combination. Metallic iron, which may find its way into the pulp during crushing operations, has no deleterious effects, its rate of dissolution being scarcely noticeable as compared with gold and silver. Limonite, $2\text{Fe}_2\text{O}_3 \cdot 3\text{H}_2\text{O}$, the soft, yellow oxide of iron, often found in oxidized ores, is scarcely acted upon by cyanide solutions. Pyrite, the most common iron mineral, is slowly acted upon by cyanide. Marcasite, while not so common, is much less soluble, but readily decomposes. The compounds of iron that are most harmful are those resulting from the decomposition of the sulphides, namely, ferrous sulphate and oxide, and sulphuric acid. Cyanide is consumed by the iron entering solution as potassium ferrocyanide, $\text{K}_4\text{Fe}(\text{CN})_6$. The interference of these elements may be overcome by a water wash or by the neutralizing properties of lime.

Sulphur.—There is no direct reaction between sulphur and cyanide, but when metallic sulphides are decomposed by cyanide or alkali they form alkaline sulphides; these react with cyanide and oxygen to form thiocyanates, or with oxygen to form sulphates. They also precipitate silver under proper conditions, and possibly gold. The interference is resisted by the use of zinc, lead, or mercury salts, which form inert sulphides.

Copper.—This element has strong affinity for cyanide, upon which it will act vigorously, especially if the copper be in a fine state of division. Copper plates are easily washed bright with

weak solution of cyanide. The compounds of copper are also more or less strongly acted upon by cyanide, according to their physical condition; those that are compact, hard, and un-oxidized, as sulphides, react more slowly than the porous and less compact ones, as oxides and carbonates. Malachite and azurite are easily dissolved and even in very small amounts give trouble. In some cases, depending upon the state of combination, a few pounds of copper per ton may prohibit cyaniding by ordinary methods. Some modification of the usual method is required, such as a low strength of solution, the use of ammonia as a preliminary dissolver of copper, or a preliminary removal of the copper by concentration or by a dilute acid wash.

Lead. — Lead is strongly acted upon by cyanide, but galena, PbS, is very slowly dissolved. The cyanidation of ores containing galena ordinarily gives little trouble, and no excessive consumption of cyanide results. Trouble may arise at the zinc-boxes, owing to a large amount of lead precipitated. Finely ground concentrate containing 36 per cent galena has been successfully treated.

Arsenic. — Arsenical ores have the reputation of being refractory, but in most cases they are amenable to the cyanide process. Arsenic itself is very slowly acted upon by cyanide, but may sometimes appear in small quantities in the zinc precipitate resulting from the treatment of arsenical ores. Arsenopyrite, $FeS_2 \cdot FeAs_2$, containing 46 per cent arsenic, a silver white to

steel gray pyrite, is abundant in gold ores. Concentrate containing 65 to 72 per cent of arsenopyrite has been successfully treated. The sulphide of arsenic, AsS, as a prominent constituent of ores, has been successfully treated, as at Mercur, Utah. In most cases a good extraction can be obtained from arsenical ores, while in others special treatment is necessary, such as fine grinding, roasting, or the use of bromo-cyanogen. Usually, a high consumption of cyanide is to be expected, due to the decomposition of the sulphides. Solutions must ordinarily be well aerated to resist the strong reducing action arising from the formation of alkaline sulphides, and in some cases the removal of these sulphides by other means may be necessary.

Antimony. — Most gold ores containing antimony are impossible to treat, owing to their hardness. Antimony does not appear to react with cyanide, but the decomposition of stibnite, Sb_2S_3 , consumes cyanide by forming alkaline sulphides and sulpho-cyanides. There is a reducing reaction similar to that with arsenical ores, but it is much more pronounced, and in some cases may make the treatment impossible. With stibnite the precious metal appears to be so firmly held that it cannot be dissolved. Strongly alkaline solutions will decompose the antimony and allow the precious metal to be better liberated, but at the expense of an increased reducing and deoxidizing power, thus causing a prohibitive consumption of cyanide. The following methods of treating antimony ores have been suggested

and tried: By preliminary treatment with a hot alkaline solution; by roasting; by fine grinding, which will liberate the values, but cannot prevent the evil effect on the solution.

Tellurium. — With telluride ores the usual process gives low extraction, not because tellurium is acted upon by cyanide, but because gold is held in a chemical combination which is unaffected by cyanide under ordinary conditions. Fine grinding with increased solvent action of bromocyanide is used in Australia to extract the precious metals. In America this chemical combination is satisfactorily broken by roasting, after which the precious metal is readily dissolved. New chemical methods analogous to the use of bromocyanide are being successfully used.

Zinc. — Metallic zinc is more easily dissolved than metallic gold. The sulphide of zinc is less acted upon by cyanide than cubical pyrite. The decomposition products of sphalerite are harmful and are readily attacked by cyanide, as are the products of decomposition of other sulphides.

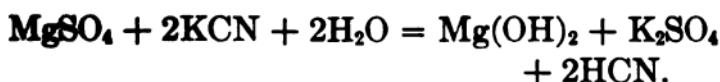
Mercury. — Metallic mercury is so slowly dissolved by cyanide that no difficulty is experienced when amalgamating in cyanide solution, or treating tailings containing mercury. Old tailings dumps often contain mercury in the form of oxide or chloride, which is easily dissolved, and later precipitated in the zinc-boxes. This small amount of mercury is beneficial in precipitating the soluble or alkaline sulphides as an insoluble and inert sulphide of mercury, HgS ; it also forms

a galvanic couple with the zinc in the zinc-boxes which assists the precipitation.

Cinnabar, HgS. — The sulphide of mercury is practically the only mineral of mercury occurring in ores of precious metals; it is unaffected by cyanide and causes no trouble.

Manganese. — Manganese is acted upon by cyanide solutions, causing a high consumption of cyanide. It is harmful in solution, but can be removed as an insoluble compound by oxidation or by the use of lime. Manganese in silver ores generally causes poor extraction, by combining the silver in a manganese compound, thus preventing the solution from dissolving it. No suitable practical method of unlocking the two has yet been devised. Good results may sometimes be secured by the following: 1. Fine grinding, followed by ordinary cyanidation or with supersolvent. 2. Roasting, followed by cyanide treatment. 3. A treatment with acid or alkali to decompose the compound.

Aluminum and Magnesium. — As sulphates, these metals sometimes cause trouble by uniting with the cyanide and liberating hydrocyanic acid, as:



Trouble is also sometimes experienced in the zinc-boxes by precipitation of these sulphates from solution. The addition of sufficient lime will avoid this, as:



Both calcium sulphate and magnesium hydroxide are insoluble and hence harmless.

Carbon and Carbonaceous Matter. — All organic matters give their share of trouble, both in dissolution and precipitation, chiefly in the following ways:

1. They are strongly reducing.
2. They consume considerable cyanide.
3. They have a tendency to reprecipitate the values in the ore.
4. They cause a vigorous evolution in the zinc-boxes.

When it is impossible to remove them, a considerable amount of lime and aeration should be used; however, lime itself may be the source of trouble if it carries carbon introduced by the lime-burning process. Graphite gives trouble mechanically, by causing a scum or froth, and chemically, by reprecipitation of the metals. Roasting would appear to be the only remedy, by oxidizing and gasifying the carbon. Weathering may render some ores suitable to treatment. (For the cause of reprecipitation see "Precipitation with Carbon" under Precipitation Methods.)

Silver Ores. — Most silver ores are amenable to the cyanide process, the extraction obtainable depending upon the nature of the mineral or silver compound. The following are amenable and a good extraction is effected by cyanide treatment: Argentite, Ag_2S , the silver sulphide; cerargerite, AgCl , the silver chloride; bromyrite, AgBr , the bromide; embolite, $\text{Ag}(\text{BrCl})$, the chlorobromide;

INFLUENCE OF OILS ON CYANIDE 33

and native silver when in a fine state of division. The following, however, do not respond readily to cyanide treatment: Proustite, Ag_3AsS_3 , the light-red ruby silver; pyrargyrite, Ag_3SbS_3 , the dark-red ruby silver; and stephanite, Ag_3SbS_4 , the brittle silver. Coarse native silver, and that occurring in galena, PbS , in tetrahedrite, $\text{Cu}_8\text{Sb}_2\text{S}_7$, and in sphalerite, ZnS , cannot be successfully treated. Silver ores in which the silver is occluded in manganese are not amenable to treatment.

Influence of Flotation Oils on Cyanide. — It does not appear that flotation oils produce any decomposition upon cyanide solutions. The following tests made by Clennell¹ would seem to prove this, the solution used in all tests containing 0.204 per cent KCN, without alkali:

No. 1.	Barrett tar.	
No. 2.	Barrett tar-oil.	
No. 3.	Crude carbolic.	
No. 4.	Crude pine oil, C. L.	
No. 5.	40-40-20 mixture	$\left\{ \begin{array}{l} 40 \text{ per cent El Paso tar.} \\ 40 \text{ per cent Barrett tar-oil.} \\ 40 \text{ per cent carbolic.} \end{array} \right.$
No. 6.	Iowa Tiger mixture	$\left\{ \begin{array}{l} 1 \text{ part Barrett tar.} \\ 3 \text{ parts carbolic creosote.} \\ 3 \text{ parts pine oil.} \end{array} \right.$

The following mixtures were agitated in the proportion shown below, and filtered. Titration of the filtrate shows practically no consumption of cyanide in any case.

¹ J. E. Clennell, *Min. & Sci. Press*, May 13, 1911.

Test No.	Solution used, c.c.	Oil used, gm.	Final solution KCN, per cent.	After flotation, CaO.
1	400	4	0.198	Trace
2	400	5	0.198	"
3	400	5	0.202	"
4	200	2	0.202	"
5	200	2	0.204	"
6	200	2	0.202	"

The proportion of oil used in these tests was much in excess of any that would be likely to occur in practice.

Effect of Size of Metallic Particles. — The rate of solution of a given amount of metal varies directly with the surface of that metal exposed. The greater the subdivision of the metal, the greater the surface exposed; consequently, the finer the particles of gold and silver the more rapidly will the total contents be dissolved by a cyanide solution. Ores that contain part or all of their values in coarse gold and silver would have to be treated by a combination of concentration followed by cyanidation, or vice versa or by amalgamation before or after cyanidation. This would give a more rapid extraction of the values.

Effect of Crushing. — The effect of crushing on some ores is to liberate the gold and silver particles almost entirely, while on others perhaps only a small portion of their surface is exposed to the action of cyanide. Other ores are of such a physical nature that if crushed to $\frac{1}{4}$ to $\frac{1}{2}$ in., all of the gold and silver can be extracted. When gold is in a fine state of division, fine grinding is

necessary, especially if the ore is so dense and compact that the solution cannot penetrate. When the gold is comparatively coarse, a face may be exposed by coarse crushing which will allow dissolution to continue slowly inward, until the entire amount of metal has been eaten out. Sulphides, especially, require fine grinding to liberate the gold they hold mechanically. Many ores contain their gold on the breaking planes, which will be liberated in the crushing process; solution then readily attacks these thin sheets of gold. Fine grinding, especially of a sliming nature, besides liberating the particles of precious metal, also breaks them down and hammers them into thin sheets so they are quickly dissolved by cyanide; whereas with a system of crushing, which merely liberates the metal, considerable time for dissolution would be required by the larger particles. Sliming in solution is especially effective for such compounds as gold with tellurium, and silver with sulphur, chlorine, etc., the ore being ground in a large volume of well-aërated solution.

The rule¹ may be given that, "the harder, denser, higher grade, more sulphuretted, and baser the ore is, the finer grained and more tightly held mechanically, combined, or alloyed the gold and silver are, the finer will be the crushing required. The softer, more leachable, oxidized, friable, porous, less base, and lower grade the ore is, the coarser and freer the gold and silver are mechanically and otherwise held, the coarser will be the crushing permissible to obtain

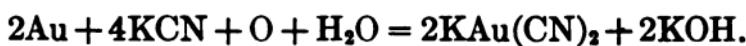
¹ "Cyanide Practice," MacFarren, 1912.

an economic extraction." This rule is modified by the effect the solution will have upon cyanides, the cost of crushing and grinding, and the expense of handling the slime, all of which increase with fine grinding.

CHAPTER IV.

DISSOLUTION OF GOLD AND SILVER IN CYANIDE SOLUTION.

Reactions. — The equation for the dissolution of gold in cyanide solution is generally accepted as follows:



This is the earliest formulation of the reaction, and is usually known, from the name of its originator, as Elsner's equation.

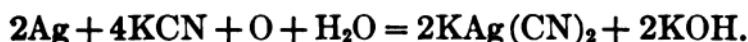
It is probable that the reaction forming $\text{KAu}(\text{CN})_2$ is not complete in one step, but takes place in the following manner:



and, finally, with excess of KCN:



Silver is dissolved in a similar way:



According to the above equations, one part of KCN should dissolve 1.51 parts of gold and 0.83 part of silver.

Oxygen, its Source and Necessity. — While the above equations would indicate that the presence of oxygen was necessary for dissolution of gold and silver, practice and experien-~~ce~~

38 DISSOLUTION OF GOLD AND SILVER

also confirmed the fact that it is beneficial. It is also known that if a sheet of gold be suspended in a solution of cyanide it will dissolve surely and steadily, indicating that dissolution is independent of oxygen.

John S. MacArthur¹ makes the following statement on this matter:

Elsner has stated that metallic gold dissolved in cyanide of potassium only in the presence of oxygen. Not having seen the original account of Elsner's researches, I am not in a position to criticise his experiments, but I never could find that the presence of oxygen was necessary to dissolve gold by itself or in ores by cyanide. If a piece of gold be immersed in a cyanide solution, so that air to act upon it would have to penetrate two to three inches of solution, the gold will dissolve in its usual slow and steady way. The equation shows that either oxygen must be absorbed or hydrogen evolved. I have seen no evidence of the former.

Since we are not concerned with the action of cyanide upon pure gold but rather its action upon gold in ores, it is interesting to see if any benefit is derived from the presence of oxygen in this case. I have made the same experiment as that described by MacArthur, treating two samples, one with free access of air, and one with boiled water, the bottle being filled to the stopper, and the stopper sealed. The extraction was the same in both cases.

The providing of oxygen when treating clean ores is not essential, but with increase of sulphides or bases in the ore the presence of oxygen

¹ "The Discovery of the Cyanide Process," *Min. & Sci. Press*, June 10, 1916.

is important, since the compounds of these metals decompose or oxidize, thereby utilizing abstracting oxygen necessary for dissolution.

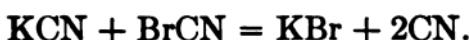
Among the best known chemical oxidizers that may be used as a source of oxygen are: Sodium peroxide, Na_2O_2 ; potassium permanganate, KMnO_4 ; and manganese dioxide, MnO_2 . They may hasten dissolution but have not been found economical. Aeration of the solution is the most practical way of supplying oxygen, and this is done in many ways.

Supersolvents. — The dissolving power of cyanide may be increased by the addition of chemicals, this result being accomplished in two ways:

1. By their oxidizing effect.
2. Through the liberation of cyanogen in a nascent state, more active in uniting with a metal which will replace the hydrogen of HCN.

The two substances most used as supersolvents are bromocyanide and mercury salts.

Bromocyanide is prepared by dissolving the commercial crystals in water, or, as at cyanide mills, by adding bromine-water to cyanide solution, when the following reaction occurs:



Cyanogen in nascent condition acts energetically on the gold as follows:

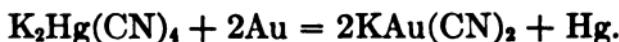


Bromocyanide does not require the prior aeration necessary in the use of the ordinary solution

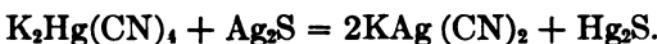
40 DISSOLUTION OF GOLD AND SILVER

is expensive, however, and has been applied to sulpho-tellurides that are unattacked, or are attacked but slowly, by plain cyanide.

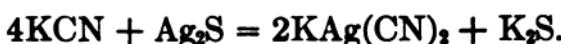
Mercury Salts. — The mercury salts that have been most used are mercurous chloride, Hg_2Cl_2 , and mercuric chloride, $HgCl_2$. The effect of these, owing to their affinity for cyanogen, is to form mercuric cyanide, $Hg(CN)_2$, or a double mercury cyanide of potassium, $K_2Hg(CN)_4$. It does this by decomposing such stable compounds as the ferrocyanides and ferricyanides, $K_4Fe(CN)_6$ and $K_3Fe(CN)_6$, in addition to the simple cyanides and easily decomposed double cyanides, thus obviating the interference of the ferrocyanides. The action of the double mercury salt is to dissolve gold without requiring oxygen, as:



With silver sulphide, the reaction is:



In this case the stable and insoluble sulphide of mercury, HgS , is formed, which is not detrimental in the manner of the alkaline sulphide, K_2S , which is formed when plain cyanide is used:



Mercury salts have been used to advantage in the treatment of silver sulphide ores.

Strength of Solution Required. — The economical strength of solution required for any given ore is a matter of experiment, and cannot be predicted. Experience has shown that both gold

STRENGTH OF SOLUTION REQUIRED 41

and silver dissolve most rapidly in solutions ranging in strength from 0.1 per cent (2 lb. per ton) to 0.25 per cent (5 lb. per ton) KCN. Within these limits the rate is almost constant, but diminishes with solutions above or below these strengths. With stronger solutions the lessened solvent effect is ascribed to the fact that the amount of oxygen soluble in a solution grows less as the cyanide content becomes greater. If plenty of oxygen, however, is supplied, a strong solution will dissolve more rapidly than a weak one; but, on the other hand, the weaker the solution the less will be the consumption of cyanide per ton of ore treated.

The strength of solution is designated in per cent of KCN, or pounds per ton of solution, as:

0.05 per cent	= 1 lb. KCN per ton of solution	
0.1 per cent	= 2 lb.	"
0.15 per cent	= 3 lb.	"
0.2 per cent	= 4 lb.	"
0.25 per cent	= 5 lb.	"
0.3 per cent	= 6 lb.	"

In leaching practice, from 0.1 to 0.02 per cent KCN are generally used, depending upon the nature of the ore. In agitation treatment, the strength ranges from 0.1 to 0.5 per cent KCN or more, depending upon the baseness of the ore. When strong solutions are used plenty of air must be provided. The consumption of cyanide is also higher on account of the increased solvent action of strong cyanide upon base metals; for this reason, as soon as the gold and silver have been extracted as far as possible, the solution should

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be withdrawn to prevent the continued action of cyanide upon base metals. In some cases, also, base metals will be taken into solution, resulting, according to mass action, in a reprecipitation of the precious metals.

Cyanicides.— Cyanicides are substances which decompose cyanide to form compounds which are not solvents of gold and silver, other than those involved in the process, as zinc. Cyanicides exert their ill effects in two ways:

1. By destroying or neutralizing the cyanide, rendering it not available for dissolving gold and silver.
2. By dissolving to such an extent that the cyanide solution becomes inactive towards both dissolution and precipitation of gold or silver.

The deleterious effects of cyanicides are counteracted in the following ways:

1. By concentration or water washing, etc.
2. By converting them into inert salts by the use of a proper quantity of lime, etc.
3. Neutralization and aeration of the pulp.
4. By passing the solution through the zinc-boxes, which often appears to clean it, or deprives the cyanicides of their power to retard dissolution.
5. By keeping the solution at a low cyanide strength, thus reducing its affinity for cyanicides, while also increasing the selective action for gold and silver.

Effect of Temperature on the Dissolution of Gold and Silver. — Practice has proved that very little is to be gained by heating the solution. It is

true, however, that in many cases laboratory tests give a higher extraction with a hot solution. If only a slight increase in extraction is obtained it is probable that this would not warrant the extra trouble and expense of heating.

CHAPTER V.

LOSSES OF CYANIDE.

Classification of Losses. — The losses of cyanide occurring during the treatment of gold and silver ores by the cyanide process may be classified as follows:¹

I. Chemical losses:

1. Reactions with the various constituents of the ore, including those introduced during its preparation for treatment.
2. Atmospheric decomposition through contact with the air, including hydrolysis.
3. Impurities in the water used.
4. Reactions occurring during precipitation.
5. Certain equipment of the plant with which the solutions may come in contact.

II. Mechanical losses:

1. Admixture of solution with residues.
2. Leakage of tanks or other containers.
3. Excess solution sent to waste.

The relative importance of these various sources of loss varies considerably, depending upon the

¹ Atmospheric Decomposition of Cyanide Solutions.
By G. H. Clevenger and Harry Morgan, *Min. & Sci. Press*,
Sept. 16, 1916.

character of the ore treated, the plant, and the manner in which it is operated. Most of the above losses have been recognized by cyaniders. The loss occurring through atmospheric decomposition, until recently, was given very little thought, and all losses were attributed to a combination of other causes.

Atmospheric Decomposition.—The chief products of atmospheric decomposition of weak cyanide solutions are:

1. Alkaline carbonate, which remains in solution as soluble sodium or potassium carbonate, depending upon which alkaline cyanide is used, in the absence of calcium hydroxide, $\text{Ca}(\text{OH})_2$; in the presence of calcium hydroxide, the carbonate, as rapidly as formed, is precipitated from solution as insoluble calcium carbonate.
2. Hydrocyanic acid, most of which passes off into the air as a gas unless there is an excess of alkaline hydroxide present.
3. Ammonia, most of which also passes off into the air as gas.

The following are conclusions drawn from the experiments conducted by H. A. White,¹ H. M. Lisle,² Dr. Moir,³ and Clevenger and Morgan:⁴

1. Loose or incomplete covering is practically useless towards preventing loss of alkali or cyanide.

¹ Cyanide Consumption on the Witwatersrand, *Jour. Met. & Min. Soc. of S. A.*, Vol. 36, pp. 24-36.

² The prevention of Hydrolysis in Cyanide Solutions, *Ibid.*, pp. 36-47.

³ *Ibid.*, Vol. 17, p. 7.

⁴ *Min. & Sci. Press*, Sept. 16, 1916.

2. With sufficient protective alkalinity and a reasonable period of treatment, the loss from atmospheric decomposition is not serious.

3. With strong solutions, the loss of KCN, both as free cyanide and total cyanide, when the available protective alkali is below 0.01 per cent NaOH, is considerable.

4. The presence of zinc enormously decreases the loss of KCN by hydrolysis, etc. The behavior of zinc is not clear, but the fact that mill solutions generally show lower cyanide consumption than fresh solutions confirms this conclusion.

5. The loss is incomparably greater with pure synthetic solutions, even with added alkali, than with ordinary solutions.

6. It is safer to omit the addition of ferrocyanide previous to the determination of protective alkali, for the reason that the ferrocyanide precipitates zinc with an accompanying increase in the alkalinity of the solution. The alkalinity thus indicated is not available for the protection of cyanide.

7. High temperature causes increased decomposition of cyanide.

8. Decomposition is reduced by the presence of real protective alkalinity.

9. Hydrolysis is of less importance than direct composition by atmospheric carbon dioxide. This statement is questionable, since it is impossible to distinguish between this reaction and the one involving the intermediate steps of hydrolysis, as the final products of the two reactions are the same.

The mill operator will naturally ask:

1. Is atmospheric decomposition of cyanide a serious factor in operating a cyanide plant?
2. What can be done to recover cyanide lost through atmospheric decomposition, and will it pay to recover it?
3. What can be done to prevent atmospheric decomposition?

These three questions can now be answered satisfactorily in the light of the South African investigations and those of Clevenger and Morgan, cited above.

1. Atmospheric decomposition may become a serious factor with cyanide solutions containing little or no protective alkalinity, but when a reasonable protective alkalinity is maintained throughout a reasonable period of treatment, the loss is not serious; the loss will vary with the time required for satisfactory extraction.

2. The cyanide lost through reactions with oxygen cannot in any circumstances be economically recovered, since the nitrogen is in the form of CNO, cyanate; under ordinary conditions this loss is small.

The cyanide lost through the reaction with CO_2 is in the form of HCN, which largely passes off with the air at the top of the tank. This can be recovered in several ways:

- a. By using closed tanks and passing all the air through scrubbing towers containing lime-water or caustic soda solution, preferably the latter.
- b. Pass the air used in agitation through scrubbing towers containing lime-water or caustic

soda, which will remove the CO_2 before introducing it into the treatment tank. In this case it would be necessary to cover the tank in order to exclude outside air.

c. By connecting agitation tanks in such a way that the air introduced into the first tank will pass through the whole series, a higher alkalinity can be maintained in the lowermost tanks of the series, or the air can be discharged into a sump solution having a higher alkalinity.

The application of any one of these systems would entail mechanical difficulties and extra expense.

3. From what has preceded, the most effective means of preventing atmospheric decomposition is to maintain a sufficiently high concentration of alkaline hydroxides in solution. This, too, like many factors in cyanidation, should be a compromise to give the best general results.



CHAPTER VI.

ALKALINITY AND LIME.

Purpose in Cyaniding. — Cyanide is protected by the use of lime and alkalinity, hence "protective alkalinity." It does this in the following ways:

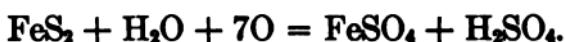
1. By combining with substances with which cyanide would otherwise unite.
2. By replacing cyanide in compounds, thus liberating it.
3. By precipitating substances that may interfere, thus keeping the solution clean and in good condition to dissolve the precious metals out of ore and precipitate them in the zinc-boxes.
4. Lime, or other alkalinity, liberates precious metals from their compounds through its solvent power on base metals. For this purpose lime acts as an alkali, and any other alkali could be used more or less advantageously for this purpose.
5. In addition to its chemical action it also acts physically on slimes, causing these light colloidal materials to be settled more rapidly.

Source of Acidity. — By the oxidation and decomposition of metallic sulphides, acids and sulphates are formed. In the case of iron pyrite the principal stages of decomposition are (From W. A. Caldecott¹):

¹ Proc. Chem., Met. and Min. Society of S. A., Vol. 4.

1. FeS_2 Iron pyrite.
2. FeS ; S..... Ferrous sulphide and sulphur.
3. FeSO_4 ; H_2SO_4 Ferrous sulphate and sulphuric acid.
4. $\text{Fe}_2(\text{SO}_4)_3$ Normal ferric sulphate.
5. $2\text{Fe}_2\text{O}_3 \cdot \text{SO}_3$ Basic ferric sulphate.
6. $\text{Fe}_2\text{O}_3 \cdot \text{XH}_2\text{O}$ Ferric hydrate.
7. Fe_2O_3 Ferric oxide.

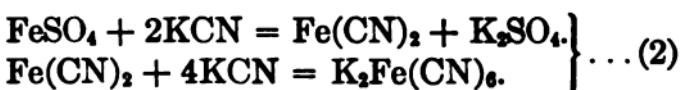
As stated under "Interfering Elements," un-oxidized iron pyrite has little deleterious effect upon cyanide, but if the sulphide is in a slightly decomposed state the following reaction will undoubtedly occur:



The sulphuric acid thus formed, unless immediately removed or neutralized, will react with KCN as follows:



Ferrous sulphate will have the following effect upon cyanide:

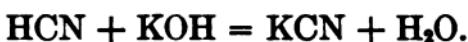


It is seen that if these two reactions are allowed to take place, the consumption of cyanide will be high, due to the formation of hydrocyanic acid, HCN, and potassium ferrocyanide, $\text{K}_2\text{Fe}(\text{CN})_6$.

Neutralization by Lime. — It is in the prevention of these reactions, especially the first, that lime finds its most useful field, its operation being as follows:



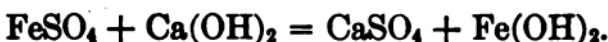
The calcium sulphate thus formed is an insoluble salt and harmless to cyanide. Also any hydrocyanic acid formed, in the presence of lime or protective alkalinity, will be generated into cyanide, as:



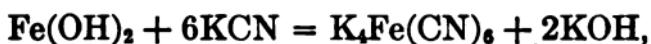
The formation of hydrocyanic acid may also be prevented by previous washing with an alkali salt, which neutralizes and removes the soluble sulphuric acid.

Referring to equation (2), further complex reactions between ferrous sulphate, FeSO_4 , and ferrocyanide, $\text{K}_2\text{Fe}(\text{CN})_6$, may take place, resulting in the formation of $\text{Fe}_4(\text{Fe}(\text{CN})_6)_3$; this gives a blue color to the solution and deposits small amounts of dark blue compound in the ore, tanks, pipes, etc. It indicates that the neutralization has been incomplete.

The action of lime upon ferrous sulphate is as follows:

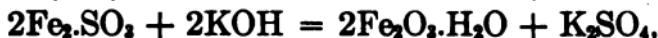
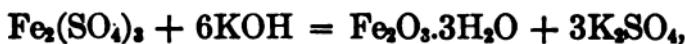


This, as can be seen from the following equation,



does not prevent the consumption of cyanide, and it is through this reaction between iron and cyanide that the greatest loss of cyanide occurs. It is prevented, in part, by efficient aeration, causing ferrous hydroxide, $\text{Fe}(\text{OH})_2$, to be oxidized to ferric oxide, Fe_2O_3 , which is harmless to cyanide.

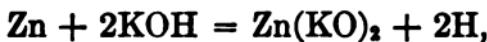
If ferrous sulphate is not removed when formed, it will oxidize to normal ferric sulphate, $2\text{Fe}(\text{SO}_4)_3$, and from this into basic ferric sulphate, $2\text{Fe}_2\text{O}_3 \cdot \text{SO}_3$, which is insoluble, but both will react with cyanide and cause a high consumption; in the presence of alkali, however, these are changed into hydrous ferric oxide, as:



which are harmless in cyanide solutions.

Importance of Alkalinity in Zinc Precipitation.

— While a protective alkalinity is usually necessary for good precipitation, at least with a weak solution, too high an alkalinity should be avoided, as it causes an excessive action on zinc, as:



resulting in an increased consumption of zinc and a production of much hydrogen, which causes an undesirable disturbance in the zinc-boxes, and poor precipitation.

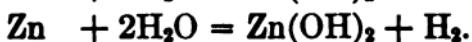
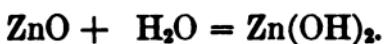
In weak cyanide solutions the zinc in solution may be considered to take the form of a hydroxide, $\text{Zn}(\text{OH})_2$, or the single zinc cyanide, $\text{Zn}(\text{CN})_2$, both of which are insoluble in water, and practically insoluble in weak cyanide solution. In strong cyanide solution the zinc will exist as zinc potassium cyanide, $\text{K}_2\text{Zn}(\text{CN})_4$, which is highly soluble, and in alkaline solutions of the alkalies as a zincate which is also soluble. With weak solutions the zinc shavings will be more or less coated with zinc hydroxide and zinc cyanide which results in poor precipitation.

White Precipitate. — White precipitate, often forming under improper conditions, consists mainly of zinc hydroxide, $Zn(OH)_2$, and zinc cyanide, $Zn(CN)_2$.

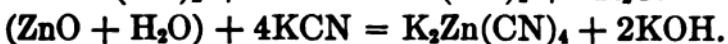
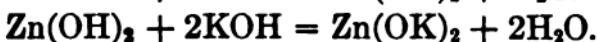
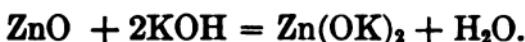
The alkaline zincate is soluble in free cyanide, as:



Zinc oxide, formed through the exposure of zinc to the air, especially if the zinc is moist, may be converted to zinc hydroxide, or the metal may be hydrolyzed directly, as:



Also, if free cyanide be absent, the oxidizing effect of the solution may cause the formation of ZnO or $Zn(OH)_2$ (*i.e.*, in the absence of alkali). The oxide or hydroxide may be changed to the zincate by an alkali or in the presence of free cyanide, as:

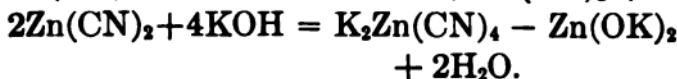


The compound $K_2Zn(CN)_4$ is insoluble in weak cyanide solutions; hence it will disassociate, as:



Zinc cyanide, $Zn(CN)_2$, is insoluble in water; thus, in weak solutions, it precipitates to form the white precipitate. This is prevented by the addition of the right quantity of alkali,

which dissolves it according to the following equations:



The formation of white precipitate may thus be prevented:

1. By the presence of a sufficient amount of protective alkalinity.
2. By precipitating in the presence of a strong cyanide solution. In most cases, neither preventative will have to be carried to extreme.

Lime vs. Caustic Soda. — It is important to note that these two alkalies cannot be used interchangeably, on account of the difference in the products formed on neutralization. Those resulting from the use of sodium or potassium are very soluble while those from lime are very insoluble. If an excess of alkali has been used, this fact may cause trouble in the zinc-boxes (to be discussed elsewhere.) The alkalies have one advantage, when such advantage is desired, in their extreme solubility; they will dissolve in an equal weight of water, while one part of CaO requires 800 parts of water. Pure lime has 1.43 times the neutralizing effect of caustic soda and is considerably cheaper.

CHAPTER VII.

TESTS BY ASSAYER AND CHEMIST.

In the control of a cyanide plant the assayer and chemist have many daily tests to perform. The following are those which may be required to control a plant already in operation:

- I. Assay of ore.
- II. Assay of cyanide solutions.
- III. Assay of zinc precipitates.
- IV. Assay of slag and mattes.
- V. Assay of bullions.

ASSAY OF ORE.

As a general rule, the percentages of extraction from various experimental treatments are calculated by assaying the tails, values in the solutions being seldom determined, except occasionally as a check. The following charges, which have been proved many times, are appropriate for the materials usually encountered.

CLEAN QUARTZ ORE.

Ore.....	0.5 A. T.
Litharge.....	2.0 A. T.
Soda, Na_2CO_3	1.0 A. T.
Flour.....	2-2.5 gr.
Light borax cover.	

56 TESTS BY ASSAYER AND CHEMIST

LOW-GRADE QUARTZ TAILING.

Ore.....	2.0 A. T.
Litharge.....	3.0 A. T.
Soda.....	2.0 A. T.
Flour.....	2.5 gr.
Heavy soda cover.	Use large crucible.

LIMESTONE OR DOLOMITE.

Ore.....	0.5 A. T.
Litharge.....	2.0 A. T.
Soda.....	1.0 A. T.
Silica.....	0.5 A. T.
Borax glass.....	1.0 A. T.
Flour.....	2-2.5 gr.
Light borax cover.	

TELLURIDE ORES.

Ore, crushed to 150 mesh.....	1.0 A. T.
Litharge.....	6.0 A. T.
Soda.....	0.5 A. T.
Borax glass.....	10 gr.
Flour.....	2-2.5 gr.
Light borax cover.	

SULPHURETS OR PYRITE-PRELIMINARY ASSAY.

Ore.....	0.25 A. T.
Litharge.....	2.0 A. T.
Soda.....	1.0 A. T.
Silica.....	0.25 A. T.
Heavy borax cover.	

Weigh the button from the above assay; then:
(Weight of button - 20) \div 0.35 = grams of
nitre to be added. Make up the charge as above,
with the addition of the necessary nitre.

Arsenical sulphides and chalcopyrite are as-
sayed in the same way except that 0.2 A. T. of ore

is taken. If the amount of copper present is large, the proportion of litharge may be increased with advantage.

LOW- OR HIGH-GRADE MATTE.

Matte.....	0.2 A. T. (or less)
Litharge.....	4.0 A. T.
Silica.....	0.75 A. T.
Soda.....	1.5 A. T.
Heavy borax cover.	

GALENA FOR SILVER.

Ore.....	0.5 A. T.
Litharge.....	2.0 A. T.
Soda.....	2.0 A. T.
Silica.....	0.25 A. T.
Flour.....	0.5-1 gr.
Light borax cover.	

ROASTED CONCENTRATE OR ORE.

(Containing much Ferric Oxide.)

Ore.....	0.5 A. T.
Litharge.....	2.0 A. T.
Soda.....	1.0 A. T.
Silica.....	0.25-0.5 A. T.
Borax glass.....	0.25 A. T.
Flour.....	3 gr.

ASSAY OF CUPELS.

(For Corrected Assay of High-grade Ore.)

Stained part of cupel.	
Litharge.....	1.0 A. T.
Soda.....	1.0 A. T.
Silica.....	0.5 A. T.
Borax glass.....	1.0 A. T.
Flour.....	2.5 gr.
Heavy borax glass cover.	

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LOW-GRADE QUARTZ TAILING.

Ore.....	2.0 A. T.
Litharge.....	3.0 A. T.
Soda.....	2.0 A. T.
Flour.....	2.5 gr.
Heavy soda cover.	Use large crucible.

LIMESTONE OR DOLOMITE.

Ore.....	0.5 A. T.
Litharge.....	2.0 A. T.
Soda.....	1.0 A. T.
Silica.....	0.5 A. T.
Borax glass.....	1.0 A. T.
Flour.....	2-2.5 gr.
Light borax cover.	

TELLURIDE ORES.

Ore, crushed to 150 mesh.....	1.0 A. T.
Litharge.....	6.0 A. T.
Soda.....	0.5 A. T.
Borax glass.....	10 gr.
Flour.....	2-2.5 gr.
Light borax cover.	

SULPHURETS OR PYRITE-PRELIMINARY ASSAY.

Ore.....	0.25 A. T.
Litharge.....	2.0 A. T.
Soda.....	1.0 A. T.
Silica.....	0.25 A. T.
Heavy borax cover.	

Weigh the button from the above assay; then:
(Weight of button - 20) \div 0.35 = grams of
nitre to be added. Make up the charge as above,
with the addition of the necessary nitre.

Arsenical sulphides and chalcopyrite are assayed in the same way except that 0.2 A. T. of ore

is taken. If the amount of copper present is large, the proportion of litharge may be increased with advantage.

LOW- OR HIGH-GRADE MATTE.

Matte.....	0.2 A. T. (or less)
Litharge.....	4.0 A. T.
Silica.....	0.75 A. T.
Soda.....	1.5 A. T.
Heavy borax cover.	

GALENA FOR SILVER.

Ore.....	0.5 A. T.
Litharge.....	2.0 A. T.
Soda.....	2.0 A. T.
Silica.....	0.25 A. T.
Flour.....	0.5-1 gr.
Light borax cover.	

ROASTED CONCENTRATE OR ORE.

(*Containing much Ferric Oxide.*)

Ore.....	0.5 A. T.
Litharge.....	2.0 A. T.
Soda.....	1.0 A. T.
Silica.....	0.25-0.5 A. T.
Borax glass.....	0.25 A. T.
Flour.....	3 gr.

ASSAY OF CUPELS.

(*For Corrected Assay of High-grade Ore.*)

Stained part of cupel.

Litharge.....	1.0 A. T.
Soda.....	1.0 A. T.
Silica.....	0.5 A. T.
Borax glass.....	1.0 A. T.
Flour.....	2.5 gr.
Heavy borax glass cover.	

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Probably the most difficult ores on which to obtain correct assays are the tellurides. The following references may be of value to those dealing with this particular class of ore.

Bailar, "Cupel Losses in Telluride Ores," *Min. Rep.*, Vol. 52, p. 44.

Borrowman, "Assay of Tellurides," *Jour. Am. Chem. Soc.*, Vol. 30, p. 1023; *Chem. News*, Vol. 98, p. 181; "Mineral Industry," Vol. 17, p. 974.

Easley and Keniston, "A Study of the Losses of Cupellation in the Presence of Tellurium," *Min. & Sci. Press*, Vol. 60, p. 563.

Fulton, "Assay of Telluride Ores," *School of Mines Quart.*, Vol. 19, p. 419; *Jour. Am. Chem. Soc.*, Vol. 20, p. 20, p. 586; *Abstracts Chem. Soc.*, Vol. 76, p. 63.

Furman, "Manual of Practical Assaying," p. 330.

Hillebrand and Allen, "Assay of Tellurides," *U.S.G.S., Bul.* No. 253.

Hollway and Pierce, "Assay of Telluride Ores," *Trans. A.I.M.E.*, Bul. 39; *Eng. & Min. Jour.*, Vol. 35, p. 619.

McLeod, "Some Experience in Assaying Telluride Ores," *Min. World*, Vol. 26, p. 682.

Smith, "Behavior of Tellurium in Assaying," *Trans. I.M.M.*, Vol. 17, p. 463.

Woodward, "Cupel Absorption with Telluride Ores," *Min. Rep.*, Vol. 52, p. 144.

ASSAY OF CYANIDE SOLUTIONS.

There are four well-known methods for determining gold and silver in cyanide solutions by the fire assay.

1. Evaporation of the solution in a tray of lead foil, followed by a cupellation of the lead.

2. Evaporation of the solution with litharge, fluxing of the litharge, fusion in the assay furnace, and cupellation.

3. Precipitation of the precious metals, filtration, incinerating the filter and its precipitates, fluxing and fusing the residue with cupellation of the lead button.

4. Precipitation of the precious metals with a large amount of lead, removing precipitate from solution, and cupelling without fusion; this is the Chiddy method.

There are several other methods of determining gold and silver in cyanide solutions, which will be discussed under the following headings:

5. Clennell's method.

6. Other methods.

7. The rapid estimation of silver by Clevenger.

1. Evaporation in Lead Tray. — Measure out 1 A. T. of solution, 29.166 c.c., from a burette or other previously graduated container, as a test tube. Pour the solution into a lead tray, made by folding lead foil about a suitable form, evaporate to dryness, fold the lead into a compact mass and cupel.

The advantage of this method lies in its simplicity. Its disadvantages are: not extremely accurate, on account of spitting; it requires considerable time; and only small amounts of solution (3 A. T.) can be taken.

2. Evaporation with Litharge. — Prepare a graniteware pan of about 10 in. diameter, or a large casserole, by greasing the inside edges slightly with vaseline; this is to prevent the solution from "creeping" during evaporation and facilitate removal of residue. Place 2 A. T. of litharge evenly on the bottom, add 10 A. T.

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(or 291.5 c.c.) of solution, and evaporate to dryness on a hot plate. Do not allow spattering to take place, nor should the residue be allowed to bake. Remove the dry residue with a spatula onto a piece of glazed paper, avoiding loss by dusting. Place 0.5 A. T. of silica in the receptacle and rub around with the fingers to remove any remaining patches of residue which may cling to the side; the silica is then added to the material on the glazed paper. Put 0.5 A. T. of soda in the bottom of a 20-gr. crucible, pour in the material on the paper, scrub out the receptacle with another 0.5 A. T. of soda, and add this to the charge. Add the proper amount of reducing agent and mix, in the crucible, with a spatula. Now add a light borax cover and assay in the usual manner.

The advantages of this method are its extreme accuracy and its suitability for the checking of other methods. Its disadvantage is that it consumes too much time to be used in the daily laboratory work.

3. Precipitation of Precious Metal, Filtration, Etc. — Numerous reliable methods of this character have been devised and were formerly used extensively in mill practice; they have now been largely superseded by the Chiddy method. The simplest and most practical of these methods is as follows: To 10 A. T. of cyanide solution, in a beaker, add 4 gr. of zinc dust and stir vigorously. Allow to stand for a few minutes and again stir. Finally add about 10 c.c. of commercial sulphuric acid and stir. After action has

ceased, add sulphuric acid, if needed, until it is in excess and the zinc is all dissolved. When no more action is shown on the addition of a small amount of sulphuric acid, filter, incinerate filter paper, and precipitate by placing in an assay crucible and setting it in the muffle of a furnace. After incinerating and cooling, add the following flux: 10 gr. of litharge, 8 gr. of sodium carbonate, 3 gr. of silica, and 1 gr. of flour. Fuse, and cupel the lead button.

4. Chiddy Method. — Place in a beaker 10 A. T. of solution, add 20 c.c. of a 20-per cent solution of lead acetate and $\frac{3}{4}$ gr. of zinc dust in the form of a water emulsion. Heat on a hot plate until almost boiling, then add 20 c.c. of c.p. HCl. Boil until the action ceases and the lead has collected as a sponge; wash twice with tap water, squeeze the water from the sponge between the fingers, wrap in lead foil, and squeeze again, cupel and weigh. The HCl used should contain no free chlorine. A zinc rod may be used instead of zinc dust with good results, if care is taken.

5. Clennell's Method. — This is based on precipitation of gold and silver from cyanide solution by the addition of copper sulphate and sulphuric acid, and recovery by either crucible or scorification methods. Place in a 400-c.c. beaker the following amounts of the substances named, in the order given, and stir after the addition of each:

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10 A. T. of solution to be assayed,
20 c.c. of a 10-per cent solution of CuSO_4 ,
20 c.c. of a 15-per cent solution of Na_2SO_4 ,
10 c.c. of a 10-per cent solution of H_2SO_4 .

Allow the mixture to settle for at least 15 minutes and then filter through a large filter paper. Place in a 20-gr. crucible one-half of the following charge, which has been previously well mixed on glazed paper:

Litharge.....	2	A. T.
Soda.....	1	A. T.
Silica.....	0.5	A. T.
Flour.....	1.5	gr.

Now insert the filter paper (undried) containing the precipitate, then pour in the remaining half of the charge, put on a light borax cover, fuse and pour as usual, and cupel.

6. Other Methods. — Another method on the Chiddy basis, described by C. E. Roadhouse,¹ is as follows:

5 to 10 A. T. of solution in a 400- to 600-c.c. squat type beaker. Add 20 to 25 c.c. of lead acetate solution and 1 to 1.5 gr. of zinc shavings, long strips preferred. Place on a hot plate, add hot water until beaker is three-quarters full, and bring to boil as rapidly as possible. Just before the boiling point is reached, add 1 to 2 c.c. of acetic acid and allow to boil a minute. The addition of acetic acid at this stage produces firm sponges that do not break up later under the

¹ "Assay of Cyanide Solutions," *Min. & Sci. Press*, May 15, 1915, p. 829.

ESTIMATION OF SILVER BY CLEVENGER 63

action of HCl on the zinc. After boiling a minute, pour off the solution closely, add 150 c.c. of a hot 10-per cent HCl solution, set back on the plate and cover with a watch glass. Rapid boiling for from 10 to 12 minutes will dissolve the zinc; then fill the beaker with cold water, decant, wash once more with cold water, squeeze the sponge into cubes, and dry. Wrap these cubes in lead foil and cupel.

7. Rapid Estimation of Silver by Clevenger.—The determination is based on the method of Volhard and is simple enough to be performed in half an hour. The method will be of great value in zinc-dust precipitation in that the proper quantity of zinc dust may be used to conform with the amount of silver contained in the solution to be precipitated.

Fill the dispensing burette (Fig. 1) with the solution to be assayed, so that the bottom of the meniscus is at the 20 A. T. mark. Turn on the suction and place two or three small filter papers upon the perforated bottom of a Gooch crucible. Moisten and carefully fit and press down around the edges so that a tight joint is made with the sides of the crucible. Measure out 5 gr. of the precipitant (good high-grade zinc dust) into a small beaker, add about 50 c.c. of distilled water, stir until pulped, and then pour into the Gooch crucible, rinsing out the last of it by means of a wash bottle. This will form a layer of precipitant in the bottom of the crucible about $\frac{1}{8}$ inch thick. Just as the last of the water is disappearing from the surface of the cake of precipitant,

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LOW-GRADE QUARTZ TAILING.

Ore.....	2.0 A. T.
Litharge.....	3.0 A. T.
Soda.....	2.0 A. T.
Flour.....	2.5 gr.
Heavy soda cover.	Use large crucible.

LIMESTONE OR DOLOMITE.

Ore.....	0.5 A. T.
Litharge.....	2.0 A. T.
Soda.....	1.0 A. T.
Silica.....	0.5 A. T.
Borax glass.....	1.0 A. T.
Flour.....	2-2.5 gr.
Light borax cover.	

TELLURIDE ORES.

Ore, crushed to 150 mesh.....	1.0 A. T.
Litharge.....	6.0 A. T.
Soda.....	0.5 A. T.
Borax glass.....	10 gr.
Flour.....	2-2.5 gr.
Light borax cover.	

SULPHURETS OR PYRITE-PRELIMINARY ASSAY.

Ore.....	0.25 A. T.
Litharge.....	2.0 A. T.
Soda.....	1.0 A. T.
Silica.....	0.25 A. T.
Heavy borax cover.	

Weigh the button from the above assay; then:
(Weight of button - 20) \div 0.35 = grams of
nitre to be added. Make up the charge as above,
with the addition of the necessary nitre.

Arsenical sulphides and chalcopyrite are as-
sayed in the same way except that 0.2 A. T. of ore

is taken. If the amount of copper present is large, the proportion of litharge may be increased with advantage.

LOW- OR HIGH-GRADE MATTE.

Matte.....	0.2 A. T. (or less)
Litharge.....	4.0 A. T.
Silica.....	0.75 A. T.
Soda.....	1.5 A. T.
Heavy borax cover.	

GALENA FOR SILVER.

Ore.....	0.5 A. T.
Litharge.....	2.0 A. T.
Soda.....	2.0 A. T.
Silica.....	0.25 A. T.
Flour.....	0.5-1 gr.
Light borax cover.	

ROASTED CONCENTRATE OR ORE.

(Containing much *Ferric Oxide*.)

Ore.....	0.5 A. T.
Litharge.....	2.0 A. T.
Soda.....	1.0 A. T.
Silica.....	0.25-0.5 A. T.
Borax glass.....	0.25 A. T.
Flour.....	3 gr.

ASSAY OF CUPELS.

(*For Corrected Assay of High-grade Ore.*)

Stained part of cupel.	
Litharge.....	1.0 A. T.
Soda.....	1.0 A. T.
Silica.....	0.5 A. T.
Borax glass.....	1.0 A. T.
Flour.....	2.5 gr.
Heavy borax glass cover.	

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nitric acid. Add distilled water to make 100 c.c., heat and titrate as directed for making a regular determination. To determine the mg. of silver to which each c.c. of solution is equivalent, divide the weight of the silver foil taken by the number of c.c. of sulpho-cyanate solution required.

Mercury interferes with this method, because its sulpho-cyanate is insoluble. Metals whose salts are colored, such as nickel, cobalt, etc., must not be present. Copper may be present up to an amount equal in weight to the silver. Copper in small amounts is a frequent constituent of mill solutions, but it is not probable that it would be precipitated sufficiently by the zinc dust to interfere with the determination.

ASSAY OF ZINC PRECIPITATE

The precipitate resulting from the clean-up of a zinc-box may be assayed by either crucible or scorification methods.

CRUCIBLE METHOD.

Precipitate.....	0.10 A. T.
Litharge.....	70 gr.
Sodium carbonate.....	5 gr.
Silica.....	5 gr.
Borax glass.....	2 gr.
Flour.....	1 gr.

SCORIFICATION METHOD.

Precipitate.....	0.10 A. T.
mixed with PbO.....	0.10 A. T.
and covered with test-lead.....	40 gr.

ASSAY OF SLAGS AND MATTES

Assay of Slags. — The charge for these depends upon whether they are acid or basic. Particular care must be taken to get a charge that will completely decompose the original slag. If this is acid the charge should aim to make a new slag more basic, and vice versa. The lead button should be from 20 to 25 gr. in weight. Many assayers simply add litharge and a reducing agent to the slag in making the fusion. This is not always desirable, for if the slag already has much litharge in it, soda, etc., may advantageously be added as the extra base in place of litharge.

Assay of Mattes. — See method under "Assay of Ores."

ASSAY OF BULLIONS

The various bullions to be assayed by the cyanider may be classified as follows:

1. Base bullion, containing from 100 to 925 parts of silver in 1000, gold in varying amounts, and a large percentage of base metals, chiefly copper, zinc, lead, etc.
2. Silver bullion, including doré bullion, fine silver bullion, and silver bullion.
3. Gold bullion.

1. Base Bullion. — The sample of bullion may be melted under charcoal and granulated in cold water, or it may be rolled out into a strip, and pieces cut out from this at intervals for the

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sample. If the bullion is free from base metals, such as copper, zinc, etc., it may be assayed by direct cupellation. In this case, four portions of 0.5 A. T. each are wrapped in about 7 gr. of sheet lead, placed in hot cupels, and cupelled with feathers.

If the bullion contains base metals which will influence the results of cupellation, four portions of either 0.5 or 1.0 A. T. are weighed out and mixed with from 30 to 50 gr. of test-lead; 1.5 gr. of borax glass and 0.5 gr. of silica are put on top of the lead and the charge scorified. The resultant buttons, which should weigh about 15 gr., are then cupelled. The scorifier slag and the cupel are reassayed and the correction is added.

2. Silver Bullion. — For silver bullion containing no gold (fine silver bullion) the three best methods for determining the silver contents are:

- (a) Fire assay; cupellation with c.p. lead.
- (b) Volumetrically, with a standard solution of salt; Gay-Lussac method.
- (c) Volumetrically, with a standard solution of sulpho-cyanate of ammonium or potassium.

For silver bullion containing gold the direct cupellation method is used. If base metals are present, it must receive one or more scorifications previous to cupellation.

a. Cupellation Method. Preliminary Assay.
— Exactly 500 mg. of bullion are weighed out on an assay balance, to save calculation, wrapped as compactly as possible in 10 gr. of sheet-lead and cupelled with the formation of litharge feathers.

The silver bead is cleaned, weighed, and parted in from 1 to 9 HNO₃ for at least 20 minutes; then, if any gold appears, heat for 5 minutes more in concentrated acid; the gold is then washed, dried, annealed, and weighed. The weight of gold found subtracted from the weight of the bead gives the approximate silver, and the weight of the bead subtracted from the weight of bullion (500 mg.) gives the base metal. This base metal is usually copper and its presence is detected by noting the coloring of the cupel.

Check Assay.—As the silver determined in this preliminary assay is low, due to absorption and volatilization, a correction of 1.2 per cent is added as an approximation, or, rather, the amount of silver found is considered as 98.8 per cent of that present, and this amount of proof silver is weighed out. To this is added, in proof gold, the amount of gold found in the preliminary assay. The difference between 500 and the sum of the corrected silver and the gold is the amount of base metal to be weighed out for the check. Since this is chiefly copper, c.p. copper foil is added. The check thus weighs 500 mg. and approximates closely the composition of the bullion. Duplicates of 500 mg. of bullion are now weighed out, and these and the check are wrapped in the proper amount of sheet-lead as determined from the table below.¹

¹ "The Manual of Fire Assaying," C. H. Fulton, p. 176.

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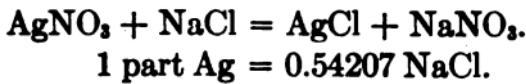
TABLE I. LEAD RATIO FOR CUPELLATION OF SILVER BULLION

Fineness in silver. Milliemees.	Amount of copper present. Milliemees.	Amount of lead for cupellation. Grams.	Ratio of lead to base metal.
			Milliemees.
1000	0	3	...
900	100	7	140-1
800	200	12	120-1
500	500	18	72-1
300	700	21	60-1

The three assays are dropped into hot cupels in an assay furnace, the check assay being in the centre. Cupellation is carried out with feathers of litharge. After the blick, the cupels are drawn to the front of the muffle and covered with extra cupels, to prevent sprouting. The beads are now cleaned, weighed, and rolled out, and parted in flasks with 1 to 9 nitric as described above. The difference between the silver actually used in the check and that found by assay is the correction to be added to the mean silver result of the two bullion assays, which should not differ by more than a millieme (0.5 point fineness).

b. Gay-Lussac Method for Silver. — This exceedingly accurate determination, which is extensively used for testing silver alloys, depends upon the precipitation of silver chloride from a nitric acid solution. The following solutions are required: (1) A standard solution of NaCl, of such strength that 100 c.c. precipitates 1 gr. of silver; hence 5.4207 gr. of c.p. NaCl are dissolved

in 1 liter of distilled water, according to the following equation:



(2) A solution of NaCl having one-tenth the strength of the former; it is made by diluting one part of the standard NaCl solution with nine parts of distilled water. (3) An acidulated solution of AgNO₃, obtained by dissolving 1 gr. of proof silver in 15 c.c. of HNO₃ (1.26 sp. gr.), and diluting with distilled water to 1000 c.c. It follows that 1 c.c. of the one-tenth NaCl solution will just precipitate the silver in 1 c.c. of the acidulated silver nitrate solution.

Standardizing Solutions. — Two portions of exactly 1002 mg. proof silver are dissolved in 15 c.c. of HNO₃ (1.26 sp. gr.), the nitrous fumes are removed by boiling, the solution is transferred to the titration bottles and water added to bring up the amount of solution to 125 c.c. A 100-c.c. pipette is then filled with normal salt solution to the mark, after washing out with salt solution to prevent dilution. The bottle containing the dissolved proof silver is placed under the pipette and the normal salt solution permitted to drain into it completely. The bottle is then violently shaken for three or four minutes, either by hand or by a mechanical agitator, and the silver chloride allowed to settle, leaving the supernatent liquid clear. If the normal solution is made up correctly, it will have precipitated just 1000 mg. of silver, leaving 2 mg. unprecipitated. One c.c.

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of the one-tenth salt solution is now added to the bottle by means of a 10-c.c. burette, which, if the solution still contains silver, gives rise to a white cloud of silver chloride. The bottle is again shaken, the precipitate allowed to settle, and another c.c. of one-tenth salt solution added. If this fails to give a precipitate, then 100.1 c.c. of normal salt solution are equivalent to 1002 mg. of silver (1 c.c. of one-tenth salt solution equals 0.1 c.c. normal salt solution). If the second addition of the one-tenth salt solution gives a precipitate, the shaking and settling are repeated, and a third and fourth, etc., addition is made until no further cloud appears. If the first addition of the one-tenth salt solution fails to give a precipitate, the normal solution contains an excess of salt, and 2 c.c. of one-tenth silver solution are now added, one of which precipitates the 1 c.c. of one-tenth salt solution added, while the other acts on the excess of salt in the solution. The one-tenth silver solution is added until no further cloud appears, in the same way as described for the one-tenth salt solution. In this way the exact strength of the normal salt solution is determined in duplicate. If it is incorrect to the extent of more than 2 points fineness either way, it is corrected by the addition of either water or salt, and standardized; when corrected, a new one-tenth solution is made up from it.

The Assay. — From a preliminary cupellation assay the quantity of bullion is taken to give as nearly as possible 1000 mg. of silver in order to make the titration as short as pos-

sible and avoid undue addition of one-tenth solution.

If mercury is present, it will interfere with the salt titration by being precipitated as Hg_2Cl_2 ; the addition of 20 c.c. of sodium acetate and a little free acetic acid to the assay will prevent the precipitation of the mercury.

Volumetric Method for Doré Bullion. — If a silver bullion contains much gold the Gay-Lussac method is generally not applicable on account of the large amount of bullion that must be taken for a sample in order to get 1 gr. of silver. The following wet method¹ will yield good results.

Fuse 0.5 gr. of the bullion with 1.5 gr. of pure cadmium under a cover of potassium cyanide in a porcelain crucible in the flame of a gas lamp. Enough cyanide must be used to cover the cadmium. Five minutes is sufficient to insure fusion. Allow to cool, place in a stream of running water, which will rapidly dissolve the cyanide and leave the alloy. Transfer this to a flask with 20 c.c. of water, add 40 c.c. of HNO_3 in installments of 10 c.c. each, while boiling for one hour; dilute to 150 c.c., add 10 c.c. of ferric-alum indicator, and titrate with a standard solution of NH_4CNS , made by dissolving 1.6 gr. of pure NH_4CNS in 1000 c.c. of distilled water. This is standardized against pure silver foil dissolved in nitric acid and diluted to 150 c.c.; 1 c.c. of the solution equals approximately 4.483 parts of silver per 1000 under the conditions described above. The indicator is a saturated

¹ "A Manual of Fire Assaying," C. H. Fulton, p. 180.

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solution of ferric-alum; appearance of the red color marks the end point. In case the bullion is very high in gold the cadmium must be increased. The parted gold is recovered from the residue in the flask.

3. Gold Bullion. — A preliminary assay as described for silver bullion is made, except that in the assay of gold bullion no determination of silver is made by cupellation; if this is to be determined the Gay-Lussac method is used. Experienced assayers can judge the approximate fineness of gold bullion by the color, and add the proper amount of silver to insure parting. In the San Francisco mint, the ratio is 2 parts of silver to 1 of gold.¹ The British Royal mint formerly used 2.75 parts of silver to 1 of gold,² but now 2 to 1. If more than 3 parts of silver to 1 of gold are present, the "cornet" of gold is apt to break up; with less than 2 parts of silver, silver is retained by the gold.

The Assay. — The final assay is prepared from the data obtained in the preliminary assay. Duplicates of 1000 milliemcs³ are run, with a check assay having a composition as near to that of the bullion as possible, as described for the cupellation assay of silver. For making up the check, proof gold and proof silver are used, and c.p. copper foil. The United States mints use various proof alloys for making up check assays:

¹ Assaying of Gold and Silver in the U. S. Mint, by John W. Pack. *Min. & Sci. Press*, LXXXVII, 317.

² Rose, *Eng. & Min. Jour.*, LXXX, 492.

³ A millieme equals 0.5 mg.

For the assay of fine gold bars (990 fineness and above), the proof alloy contains 1000 gold, 2000 silver, and 30 parts copper; for coin metal (900 parts fine), the proof alloy is 900 parts gold, 90 parts silver, and 10 parts copper; for the assay of mill bullion the proof alloy for fine gold bars is generally used.

The amount of lead required for cupellation is as follows:¹

TABLE II. LEAD RATIO FOR CUPELLATION OF GOLD BULLION

Amount of gold per 1000 parts. Milligrams.	Amount of lead. Grams.	Ratio of lead to copper (base metal present).
916.6	8.0	96-1
866	9.15	68-1
770	14.75	64-1
666	16.00	48-1
546	17.50	38-1
333	18.00	25-1

To the 1000-millieme duplicates of bullion, enough silver is added to bring the ratio of silver to gold to 2 to 1, and they are then wrapped in the proper amount of c.p. sheet-lead. The check is made up as indicated by the preliminary assay, and the three assays are cupelled as described for the assay of silver bullion; the three beads are then treated and parted. The two bullion assays should not differ by more than 0.25 millieme. The correction as indicated by the

¹ Rose, "Metallurgy of Gold," 1902, p. 494.

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check should then be applied, whether this be plus or minus. The difference between the fine gold in the check and that obtained by the assay of the check is the surcharge. This surcharge will usually be zero for a bullion of about 700 to 800 fine; with finer bullions there will be a "plus surcharge" and with less fine bullions a "minus surcharge." The plus surcharge will be subtracted and the minus surcharge added.

CHAPTER VIII.

METALLURGICAL TESTS.

When testing an ore for economical treatment by the cyanide process a great many tests may have to be conducted. Those usually made are as follows:

I. Physical Tests.

1. Screen analysis.
2. Concentration.
3. Amalgamation.

II. Acidity, Alkalinity, and Cyanide Strength.

1. Soluble and total acidity.
2. Alkalinity, and analysis of lime.
3. Free and total cyanide.

III. Extraction Tests.

1. Bottle tests.
2. Air agitation.
3. Ball-mill tests.
4. Tests with supersolvents.
5. Roasting preliminary to cyanidation.
6. Percolation.

PHYSICAL TESTS.

Physical Nature of the Ore. — When a new ore is to be tested for the cyanide process, presupposing that a representative sample both in

grade and chemical composition has been taken, the first thing to do is to investigate its physical nature and characteristics. An investigation by means of panning, assaying, microscope, etc., will indicate in no small measure the treatment that will probably be required. The following points should be noted:

1. For a soft and porous ore which solution can easily penetrate, coarse crushing only will be required.
2. If the ore is hard and dense, the solution cannot penetrate it and fine crushing will be required to liberate the metal to the solvent action of the cyanide.
3. The metal may lie on the breaking planes of the ore or on the faces of the crystals, in which case extremely fine crushing will not be required to expose it; if contained in the crystals, fine crushing will be necessary.
4. Metal present in coarse grains will have to be removed by amalgamation or concentration, as the economical time for dissolving them by cyanide solutions will be exceeded. A suitable grinding machine may flatten out these grains so that they would be dissolved quite readily.
5. On the other hand, if the metal is present in a fine state of division, cyanide solution of low strength will dissolve it quickly.
6. Limonite, kaolin, alunite, etc., cause trouble by their colloidal nature, and are slow settling; the leaf filter must be resorted to.
7. If sulphides are present, the question arises as to their removal before or after treatment,

if they carry considerable values. If not, they may be finely ground, provided the cyanide consumption is not too high.

8. Telluride ores will require roasting, or treatment with some supersolvent, as bromocyanide. Flotation may recover much of the value in this case.

9. Ores containing copper will require concentration or leaching with dilute sulphuric acid before treatment with cyanide. Other copper ores consume little cyanide if treated with a low-strength solution, increasing the selective action on gold and silver.

10. Antimony and, to some extent, manganese ores may require considerable aeration, etc.

11. Clean siliceous ores are easily treated and require little lime or alkalinity, while base ores will require a large amount.

Screen Analysis. — A sizing test on an ore, previous to other tests, may give information as to the assay value of the several sizes; however, ordinary extraction tests are not made on the various sizes until the preliminary tests are well worked out.

There are three general methods of making sizing tests: (1) Dry; (2) wet; (3) wet and dry combination.

Dry Sizing. — This is the most rapid method but it is always accompanied by a small loss from dusting, etc. To overcome this, there are three methods in common practice. They are:

(a) Start the test with a sample from 1 to 3 gr. overweight, say 1002 gr. By experience this

quantity can be very definitely determined so that the final totals will show 100, 250, 500, or 1000 gr. as required. The larger the sample the greater the loss will be, so that if a total of 1000 gr. be desired, it would be well to start with 1003 gr.

(b) Divide the loss in proportion to the weights of the different products.

(c) Assume that all the loss is in the fines, that is, finer than 200 mesh.

To save time, it is well to place the various sieves to be used in one nest, the coarsest at the top. The sieves can be roughly shaken in this manner, and as the product works down to the finer sieves, these will not be overloaded, and a quick separation is obtained. Mechanical shakers for a nest of screens are obtainable. After shaking the sieves roughly, each sieve should then be shaken separately over a pan or paper to make sure that the screening is complete; this may require from four to ten minutes, depending upon the fineness and quantity of the product, and the area of screening surface in the sieve. For ordinary laboratory work, a light washer may be placed in each sieve to help keep the mesh free and the products stirred up so that they will pass through more rapidly, but for delicate screening, this use of washers is objectionable.

Wet Sizing. — Many authorities consider it better practice, whenever possible, to make all screen analyses by the wet method. This is done by first removing all clay and slimy matter

by repeated washings and decantations, taking care not to decant any of the sands coarser than 200 mesh and then screening each size individually under water.

Wet and Dry Sizing. — Wash and decant slimes as above, then wet-screen the de-slimed sands on some intermediate screen, say 65 mesh; finally wet-screen the material below 65 mesh on the 200-mesh screen. In this way, the amount of material to be handled on the 200-mesh is much reduced. Dry both the +65-mesh and +200-mesh sands at a low temperature, and complete the screen analysis on these sands in the usual manner for dry testing. Combine all the -200-mesh screenings with the decanted slimes, settle, decant the clear water, and evaporate to dryness.

Forms for Recording Screen Analyses. — The following system is taken from catalogue No. 36 of W. S. Tyler Co., Cleveland, Ohio. It is suitable for any system of screens (Fig. 2). From this table a percentage-weight curve and assay-value curve can be drawn directly by plotting mesh against weight and mesh against assay value. The column "Diameter Wire, Inches" may be omitted.

The same system, with slight modifications, is suitable for sizing tests with cyanide extraction tests on each mesh (Fig. 3). From this record the value of gold and silver can each be plotted against mesh and also the percentage extraction.

Graphic Illustration of Sizing Tests. (From catalogue No. 36, W. S. Tyler Co.) — Referring

Indicate the screen crushed through and also first retaining screen.	Screen scale ratio 1.414.		Weights.		Per cent cumulative weights.	Assays.	Contents.	% of total contents.
	Openings.	Mesh.	Sample weights, grams.	Per cent.				
	Inches.	Millimeters.	Diameter wire, inches.					
	1.050	26.67	0.149					
	0.742	18.85	0.135					
	0.526	13.33	0.105					
	0.371	9.423	0.092					
	0.263	6.650	3	0.070				
	0.185	4.690	4	0.065				
	0.131	3.237	6	0.036				
All pass	0.093	2.362	8	0.032				
Retained on	0.065	1.651	10	0.035	70.0	14.0	14.0	
"	0.046	1.168	14	0.025	60.0	12.0	26.0	
"	0.0328	0.883	20	0.0172	50.0	10.0	36.0	
"	0.0232	0.589	28	0.0125	55.0	11.0	47.0	
"	0.0164	0.417	35	0.0122	50.0	10.0	57.0	
"	0.0116	0.295	48	0.0092	35.0	7.0	64.0	
"	0.0082	0.208	65	0.0072	30.0	6.0	70.0	
"	0.0058	0.147	100	0.0042	32.5	6.5	76.5	
"	0.0041	0.104	150	0.0026	20.0	4.0	80.5	
"	0.0029	0.074	200	0.0021	17.5	3.5	84.0	
Pass	0.0029	0.074	200	0.0021	80.0	16.0	100.0	
					Totals	500.0	100.0	

FIG. 2. Form for Recording Sizing Test.

ILLUSTRATION OF SIZING TESTS

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Screen scale ratio 1.414.			Weights.			Assay value.			Per cent extraction.		
Openings,	Mesh.	Diameter wire, inches.	Sample weights.	Per cent.	Per cent cumulative weights,	Gold.	Gold.	Silver.	Gold.	Silver.	
Inches.	Millimeters.		Oz.	\$	Oz.	Oz.	\$	Oz.	\$	Oz.	\$
Indicate the screen crushed through and also first retaining screen.											
1.050	26.67	0.149									
0.742	18.85	0.135									
0.525	13.33	0.105									
0.371	9.423	0.092									
0.263	6.680	3	0.070								
0.185	4.699	4	0.065								
0.131	3.327	6	0.036								
0.093	2.362	8	0.032								
All pass.			grams								
Retained on.			70.0			14.0		14.0			
0.65	1.651	10	0.035								
0.46	1.163	14	0.025			60.0		12.0			
0.328	0.833	20	0.0172			50.0		10.0			
0.232	0.589	28	0.0125			55.0		11.0			
0.164	0.417	35	0.0123			50.0		10.0			
0.116	0.295	48	0.0092			35.0		7.0			
0.082	0.208	65	0.0073			30.0		6.0			
0.058	0.147	100	0.0042			32.5		6.5			
0.041	0.104	150	0.0026			20.0		4.0			
0.029	0.074	200	0.0021			17.5		3.5			
Pass.	0.0029	0.074	200	0.0021		80.0		16.0			
					Totals	500.0		100.0			

FIG. 3. Form for Cyanide Extraction Tests.

TABLE I. LEA
TION OF

Fineness in silver.	Amount copper present.
Millieme.	Millieme.
1000	0
900	100
800	200
500	500
300	700

The three assays in an assay furnace centre. Cupellation of litharge. After the lead is cast to the front of the crucible, the cupels, to prevent spattering, are cleaned, weighed, and placed in flasks with 1 to 9 ml. of water. The difference between the water level in the check and that found in the flask is to be added to the result of the bullion assays, which are less than a millieme (0.5 mg.).

b. Gay-Lussac 1. An exceedingly accurate method extensively used for testing upon the precipitation of the nitric acid solution. The requirements are: (1) A standard solution of such strength that 1 ml. contains 1 g. of silver; hence 5.4207 g.

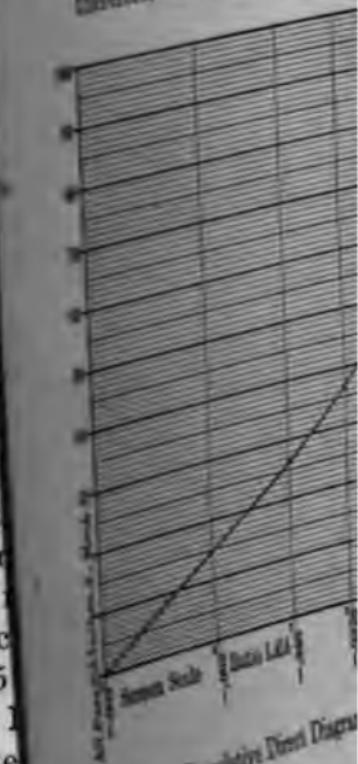


FIG. 4. Cumulative Plot Diagram

terminates at the upper right-hand corner, and drawn between these two points is the line of perfect crushing. The plotted curve approaches the actual crushing curve.

ILLUSTRATION OF SIZING TESTS 85

been reduced with rolls until all passes the 0.093-in. (8-mesh) screen. The table, Fig. 2, gives the data from which this curve has been plotted.

The intersection of the 200-mesh vertical with the curve shows that the cumulative (or total) percentage of material held on this size was 84 per cent, leaving 16 per cent as the amount passing through, which is expressed by the portion of the curve lying to the right beyond the 200-mesh intersection and terminating at the upper right-hand zero.

Cumulative Logarithmic Plot.—A very ingenious and satisfactory way to plot a curve, covering a wide range of sizes, is by the use of a method in which equal distances on the horizontal scale represent equal ratios of diameters, thereby compressing the curve for the coarse sizes and extending it for the small sizes (the opposite being true in the direct plot). This result is secured automatically by plotting the logarithms of the diameter of the opening on the horizontal scale (Fig. 5).

There are many advantages in the graphic method of illustrating the data obtained by a screen analysis. Plotted curves of the crushing by different methods or from two competing machines, for instance, express the differences in a more striking and concise manner than the tabulated data. Of the several methods of plotting these curves, the "cumulative direct plot" and the "cumulative logarithmic plot" are the two most valuable and generally used. The

curves in either plan are plotted by marking the cumulative percentages of the materials which remain on the sieves on the vertical lines corresponding to the several openings on the horizontal scale of the diagram. After drawing in

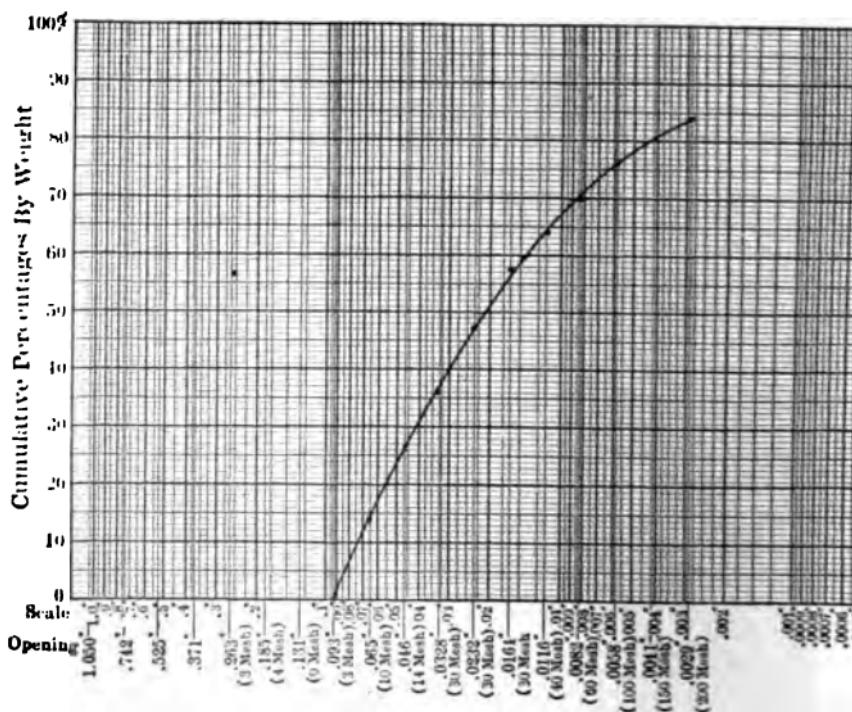


FIG. 5. Cumulative Logarithmic Diagram for Screen Analysis.

the curve, the percentage remaining on any set of openings, other than those of the testing sieves used, can be found by interpolation, and in this way the redistribution of the same material by any assumed set of openings can be determined.

The difference between a direct and a logarithmic diagram is that in the direct plot the sieve

openings are laid to an arithmetical scale, while in the logarithmic plot they are laid down to the logarithmic scale. In the arithmetical scale the horizontal distances between every two vertical lines increase or diminish in the same ratio as that in which the screen openings do, and in the logarithmic scale the screen openings are plotted to the logarithm of the diameter of the opening (the ratio between the sizes being a constant); the horizontal scale of screen openings will be one of equal spaces.

Cumulative Direct Plot. — This form of plotting is unique, since all crushings, to whatever size, are expressed on diagrams of the same uniform dimension and having the same length of base line. This renders a comparison of the curves from various crushings more comprehensive than when diagrams with varying length of base line are used.

Screen Scale. — It is well-known that the word "mesh," in a technical sense, is meaningless unless the diameter of the wire is also given, whence the opening is determined. In the Tyler standard screen scale the term mesh has been made secondary and its use should be discontinued as far as possible.

The ratio between the different sizes of the screen scale has been taken as $\sqrt{2} = 1.414$, as recommended by Rittinger in his work on ore dressing. This makes the area of each successive opening in the scale just double that of the next finer or half that of the next coarser screen. In other words, the diameters of the successive

have a constant ratio of 1.414 while the areas of the successive openings have a constant ratio of 2. Where closer sizing is required in the finer openings, the ratio as recommended by Prof. Richards may be used; here the openings increase in the ratio of the fourth root of 2 or 1.189.

For many ore testing purposes an arbitrary screen scale is chosen, depending upon the value of the ore; as, 10, 20, 30, 40, 60, 80, 100, 150, and 200 mesh. In plotting the results from such a set, the ordinary method of plotting weight or percentage of material on the various screens against mesh is used.

Concentration. — The ore should be carefully crushed, to a degree of fineness sufficient to liberate all the heavy minerals, and concentrated by panning off the light material. The resulting concentrate is assayed, and if it appears to carry the bulk of the gold and silver it should receive much consideration. It should be tested by amalgamation to determine the percentage of extraction; and by preliminary roasting followed by cyanide treatment, either leaching or fine grinding, etc. Fine grinding in cyanide solution should be tried. The concentrate may be economically shipped to a lead or copper smelter.

Amalgamation. — Amalgamation tests, in case the ore contains free gold, should be made before screen analysis to remove the metallic gold; if not done, the screen analysis is of little value. Amalgamation tests may be made on all gold and silver ores, several methods being available: (1) Agitating in a bottle. (2) Grinding in a

porcelain mortar. (3) Panning in a copper-bottomed gold pan.

Agitating in a Bottle.—Put 6 to 8 A. T. of ore, crushed to the desired mesh (say 100 mesh), in a large glass bottle with sufficient water to make a fluid pulp. Add from a burette 2 c.c. of mercury.¹ The bottle should be rolled on a piece of cloth or on a rolling agitator (Fig. 7), to cause slight agitation of its contents; violent agitation should be avoided lest the mercury be broken up into fine particles in which state it cannot be removed from the pulp by panning. After several hours the contents are washed out into a gold pan, in which it is panned and re-panned until the mercury is free from pulp and sands. The tailings are decanted, dried, and assayed. The difference in the assay of heads and tails represents the percentage extraction.

Grinding in a Porcelain Mortar.—The ore, in the form of a thin pulp, is ground with mercury in a porcelain mortar for a given length of time, say one-half hour. The contents of the mortar are washed into a gold pan where the mercury is panned free from gangue. The mercury may be assayed by dissolving in nitric acid, or by volatilizing and scorifying, or the tails are assayed and the extraction calculated.

Panning in a Copper-bottomed Gold Pan.—By this method the results obtained in practice can be more nearly duplicated. The copper bot-

¹ Mercury should be assayed for gold by dissolving in nitric acid, washing, igniting, and weighing the remaining gold bead.

tom is first brightened by scouring with fine sand and finally with a weak solution of potassium cyanide. When perfectly clean, rub in, with a brush, as much mercury as it will hold without "flowing" (this is done in a weak cyanide solution). When no more mercury will unite with the copper the excess of mercury is brushed out and the pan washed under a tap to remove the last trace of cyanide. The ore is now added in the form of a thin pulp and a panning motion given to it. In from one-half to one hour's time the extraction will be as complete as possible. Remove pulp to another pan, allow to settle, decant, dry, and assay. As before, the difference between the assay of heads and tails will give the percentage extraction.

With this method the material can be concentrated in the same tests. Laboratory tests, as a rule, do not give so high an extraction as will be obtained in actual practice. This may be due to the effect of dry crushing in the laboratory, during which the gold particles become filmed with dirt and hence will not amalgamate with mercury. Also, the individual gold particles have a better chance of coming into contact with mercury while passing in a thin layer over amalgamated plates. Ordinarily, too, the plates in a mill are in better working condition.

ACIDITY, ALKALINITY, AND CYANIDE STRENGTH.

Acidity. — To determine the amount of lime or other neutralizer required, the following tests must be made: (1) Soluble acidity; (2) total acidity; (3) latent acidity.

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Soluble Acidity. — Place 26 gr. of ore, ground to 100 mesh, in a clean, dry, agitation bottle with 100 c.c. of distilled water and agitate from one-half to one hour. Allow to settle for a few minutes, then decant the clear supernatant liquid into a filter and collect the filtrate in a clean dry beaker. Do not wash the residue or otherwise dilute the filtrate. Measure 50 c.c. of filtrate into an Erlenmeyer flask, add a few drops of phenolphthalein, and titrate with standard alkali solution (see under Titration for Alkalinity, Acid, and Free Cyanide). Care must be taken not to over titrate, because the amount of soluble acidity in most ores is very small. If, however, this should happen, the solution may be titrated back with standard acid. The number of cubic centimeters of alkali used measures the amount of soluble acidity.

Total Acidity. — Place 26 gr. of ore (100 mesh) in a clean agitator bottle with 100 c.c. of standard alkali solution. Agitate for at least six hours. Allow to settle, decant the clear liquid, and collect the filtrate in a clean, dry beaker. Titrate 50 c.c. of the filtrate with standard acid solution. The difference between 50 c.c. and the number of cubic centimeters of standard acid used equals the number of cubic centimeters of standard alkali neutralized by the total acidity of the ore, and hence the number of pounds of lime to be added per ton.

Latent Acidity. — This is found by subtracting the soluble acidity from the total acidity.

Method of Using Lime and Cyanide. — The weight of lime to be added to each test may be

determined by the following formula: Grams of lime = $0.0005 A \times B$, where A = weight of sample in grams; B = pounds of lime needed per ton of ore.

Before commencing bottle or agitation tests it is well to prepare a small carboy of strong cyanide solution, say 40 lb. per ton (2 per cent). From this, solutions of the required strength can be prepared by the following formula: $(Q \times \% W) \div \% S = \text{c.c. strong solution}$; and $Q - \text{c.c. strong solution} = \text{c.c. water to add}$; where Q = c.c. of solution desired, W = weak solution, S = strong solution. The strength of the strong solution should be accurately known, and the amounts of strong solution and water should be accurately measured or results of only approximate correctness can be expected. It is well to check each solution after the above preparation by titration with silver nitrate.

When weak solution is to be strengthened by the addition of strong solution, the strength of each being known, use the following formula:

$$x = \frac{A (a - c)}{b - c}$$

and $A - x$ = c.c. strong solution to add; where A = c.c. of solution desired, a = $\%$ KCN in solution desired, x = c.c. of weak solution in mixture, b = $\%$ KCN in weak solution, c = $\%$ KCN in strong solution.

Grinding the Ore. — The average size to which an ore is now crushed for cyanide treatment is probably 100 mesh, and this is taken as a con-

METHOD OF USING LIME AND CYANIDE 93

venient starting point. About 1.5 kg. of ore to be tested (10 mesh) is weighed out and screened through a 100-mesh screen, the oversize being further reduced in a disc pulverizer to pass this screen.

In regard to the grinding, care should be taken that the results reported are obtained on ore



FIG. 6. Laboratory Disc Pulverizer.

which is actually of the same fineness as stated in the report. Much more favorable results will generally be obtained on 150-mesh material than on 100-mesh. In practice, fine grinding is expensive and the metallurgist should know the exact fineness of the ore so that he may balance the cost of grinding against the increased extra-

tion. The method of grinding the ore is of no small importance, as the amount of fine material produced in crushing given ore through, say, 100 mesh, depends upon the type of grinding machine used, and unless care is taken it will not correspond to that which may be obtained in practice. Ordinarily, fine grinding in the laboratory is accomplished by the disc pulverizer, Fig. 6, which should receive careful adjustment and should not make too great a reduction in size in one pass. Two or three passes, at least, are required to produce 100-mesh material from 10-mesh feed.

Agitation. — For this purpose various agitators are used (see Figs. 7, 8, 9, 10). The

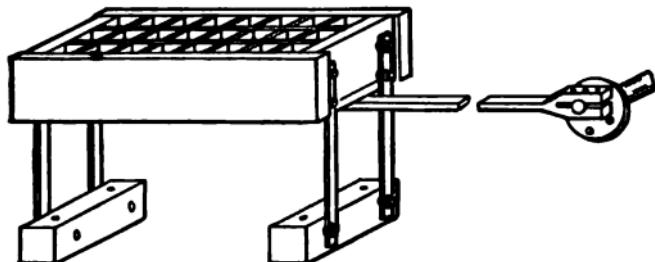


FIG. 7. Homestake Agitator for Amalgamation and Cyanide Tests.

apparatus shown in Fig. 10 has been used by the author for some time, and has been found very simple and convenient, especially when a large number of agitations are to be made. The roll agitator¹ (Fig. 8) is cumbersome and awkward, requiring much space and consumes much power per agitation. Fig. 7² illustrates an agitator used

¹ Used at South Dakota School of Mines.

² See Fulton, "A Text Book of Fire Assaying."



AGITATION

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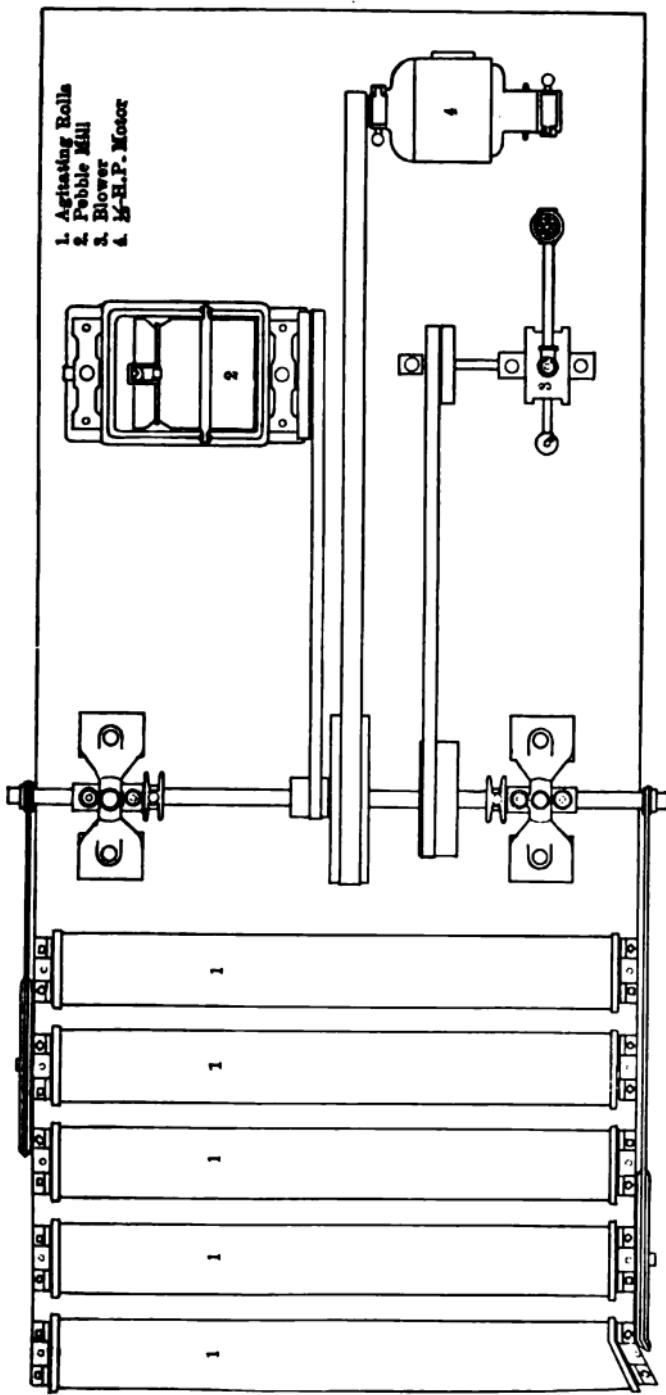


FIG. 8. Arrangement of Agitator, Pebble-mill, and Blower.

by the Homestake Mining Company; it has advantages, but I have found it less convenient than the one in Fig. 10. The agitator shown in Fig. 9 is suitable for making a less number of agitations, and especially if it is desired that the bottle be open during agitation; I find that this gives but little aeration and the extraction is seldom higher than with closed bottles.

For the agitation tests 100 gr. of ore is placed in each bottle (ordinary fruit jars of pint size

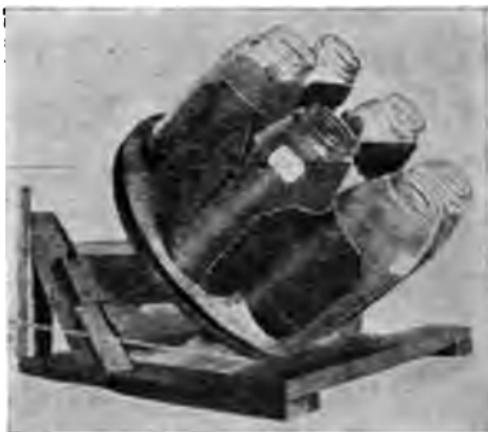


FIG. 9. Bottle Agitating Device.

are satisfactory) with the required amount of lime or other alkali and cyanide solution. Every bottle should be labeled with a serial number so that no mistake can occur through getting bottles mixed. The time of starting each test should be so arranged that it will be convenient to take off the bottles at the expiration of the period of agitation. Just before placing the bottles on the agitator the cyanide solution should be added and



AGITATION

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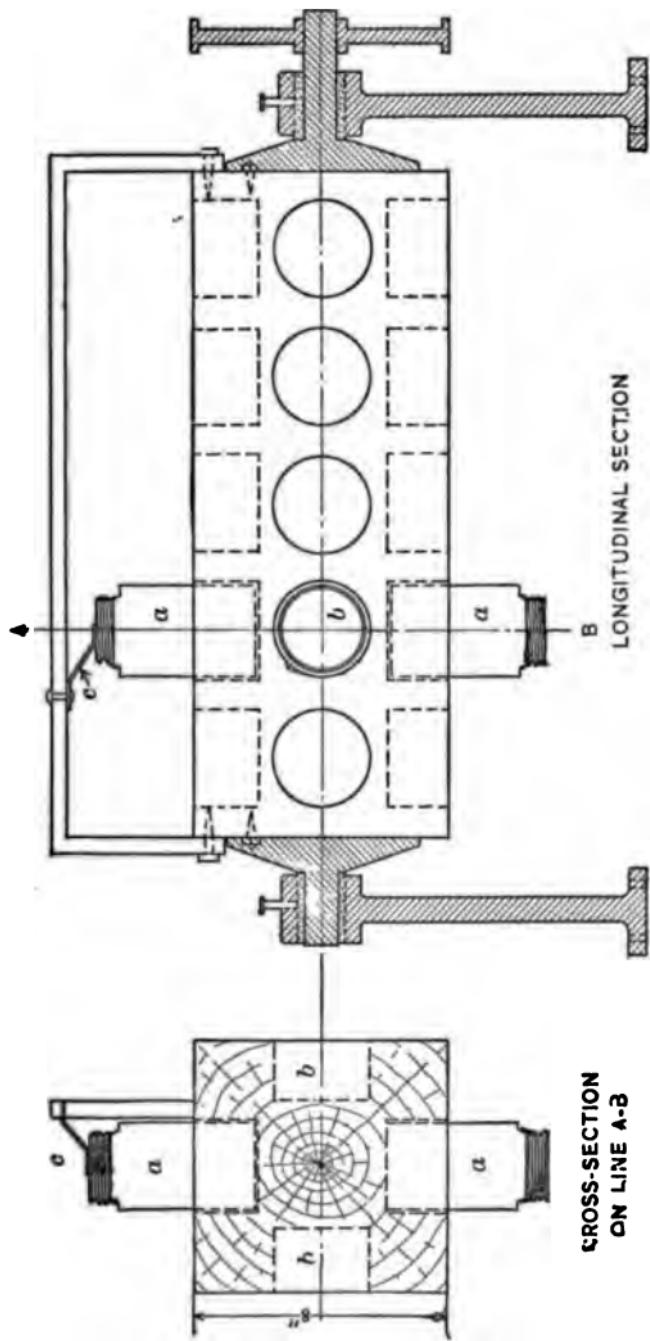


Fig. 10. The Author's Bottle Agitating Device.

the bottles well stoppered. After the bottles have been agitated for the required length of time, remove from the agitator and allow to settle, then take 50 c.c. of the clear supernatant liquid from each bottle with a pipette, place in Erlenmeyer flask, and titrate as explained under "Titration for Alkalinity, Acid, and Free Cyanide."

Washing Residues.

— Wash all contents of the bottle into a Buchner's filtering funnel, Fig. 11, placed in a filtering flask, connected with a suction of some type. An ordinary filter paper on the perforated plate of the funnel makes a very good filtering medium. The paper should be placed dry, and the suction turned on; then with

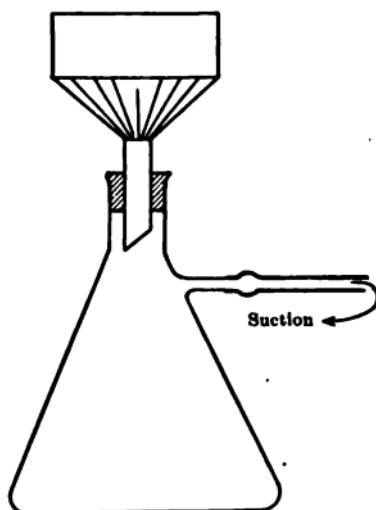


FIG. 11. Suction Filter for Residues.

a water bottle wet down the paper; as soon as a vacuum is formed pour in the contents of the agitated bottle rapidly; this prevents the light slimy material from forming the first layer upon the filter paper, filling up the small pores, and preventing rapid filtration.

The charge on the filter paper should be thoroughly washed several times with clear lime water. This serves the double purpose of washing all dissolved values from the residues, and assures

the removal of rich sulphide on the bottom of the bottle. The use of lime water instead of tap water is to be preferred, as it serves to prevent a reverse action, by which the values already in solution might be precipitated in the ore, and prevent their removal by washing. When washing is complete and the residue has been sucked dry, remove funnel with residue. The residue is conveniently removed by inverting the funnel and blowing gently into the end. The hand should be placed between the mouth and end of the funnel to avoid the possibility of being poisoned by cyanide. The residue is blown into a small sample pan, placed on a hot plate, dried, mixed, and assayed.

Alkalinity of the Solution. — Lime is used almost universally as a source of alkali in cyanide work. It is therefore taken as the standard, although there are always other alkalies in the solution, such as NaOH and KOH. Lime is soluble in water to the extent of 0.15 per cent, and $1/100$ of the quantity of lime required to make a saturated solution is called a "point of alkalinity." A solution containing one-half the lime it is able to dissolve is said to have 50 points of alkalinity. Since other alkalies are always present in working solutions, it is possible that the titration may show over 100 points of alkalinity, but this is seldom the case.

In practice, the amount of alkalinity to insure the best working conditions is determined by experience, and is thereafter kept as nearly as possible at that point. The value of lime for



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Fig. 1

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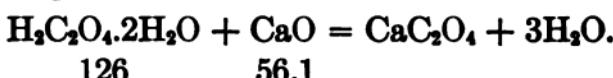


then replace the stopper and shake again. When the color returns, if it is due to lime it will be a bright, vivid pink, and the addition of perhaps 0.5 c.c. of solution will be necessary to discharge this color; but if, on shaking again, the color is a faint, weak pink returning slowly, this is the endpoint for the lime, and indicates that the magnesia, if present, is asserting itself.

At all times during the addition of the oxalic acid the flask should be violently shaken, being careful not to allow any of the solution to splash out, so that the calcium oxide will dissolve readily. In nearly every case of titrating a high-grade lime, the pink color will remain vivid nearly to the finish, which shows that calcium oxide is rapidly soluble. If a completed titration is allowed to stand for from 15 to 30 minutes, the pink color will return and show as brightly as in the beginning. The reading of the burette is in percentage of calcium oxide. The solutions necessary are: Oxalic acid, 14.6068 gr. of pure crystals dissolved in enough water to make a liter of solution. Phenolphthalein, 0.5 gr. dissolved in 50 c.c. of alcohol and 50 c.c. of water.

Titration for Alkalinity.—A standard solution of oxalic acid, $H_2C_2O_4 \cdot 2H_2O$, is required. This is prepared by dissolving 14.6068 gr. of pure oxalic acid in a little distilled water and diluting to 1 liter; 10 c.c. of this solution is required to react with 50 c.c. of saturated lime solution; hence, each $\frac{1}{10}$ c.c. of standard solution used in titrating 50 c.c. of lime solution represents a "point of alkalinity."

In 50 c.c. of saturated lime solution there is $0.0013 \times 50 = 0.065$ gr. CaO. The lime necessary to make 1 point of alkalinity in this case is 0.00065 gr. Oxalic acid combines with lime according to:



The amount of oxalic acid required to combine with 0.00065 gr. CaO is given by x in the proportion, 126 : 56.1 : : x : 0.00065; whence

$$x = 0.00146068.$$

This weight of acid is to be contained in 0.1 c.c. of standard solution; hence, 1 liter of such solution will contain 14.6068 gr. of oxalic acid.

The following method of determining protective alkalinity is based on a method first suggested by Clennell: Titrate 50 c.c. of clear solution in an Erlenmeyer flask with standard AgNO_3 solution (see below) to the first white turbidity. The number of cubic centimeters divided by 5 gives the cyanide strength in pounds per ton. Then add a few drops of phenolphthalein and titrate with standard oxalic acid solution until the color is discharged. The number of tenths of one c.c. of standard solution represents the points of alkalinity. An objection to the above method has been raised on the ground that in the presence of zinc compounds the results are inaccurate. The results, however, are sufficiently accurate for most purposes.

Titration for Acidity. — A standard alkali solution of equal strength to the standard acid

may be prepared as follows: The reaction between oxalic acid and caustic soda is represented by:



in which 126 parts of oxalic acid neutralizes 80 parts of sodium hydroxide. The weight of sodium hydroxide which should be contained in a liter of standard alkali is found therefore from the proportion: 126 : 80 :: 14.6068 : x , whence $x = 9.274$.

In weighing out the NaOH (which is seldom pure) it is best to take 9.3 gr. and dissolve in 900 c.c. of distilled water, making preliminary titrations against standard acid, and calculating the dilution necessary. For example, suppose that it is found that 10 c.c. of the alkali solution required 11.2 c.c. of the standard acid. Therefore to each 10 c.c. of the alkali remaining of the 900 c.c. there must be added 1.2 c.c. of water. There remains $900 - 10 = 890$ c.c.; the water to be added therefore equals $(890 \div 10) \times 1.2 = 106.8$ c.c. The alkali solution, if kept in a well-stoppered bottle, will vary but little.

To determine the acidity of an ore, 26 gr. are agitated with 100 c.c. of water and filtered; the acidity of 50 c.c. of the filtrate is then neutralized with standard alkali solution. The 50 c.c. titrated contains the acid from 13 gr. of ore; hence, $13 \times 0.005^1 = 0.0065$ gr. of lime to be

¹ If lime is added at the rate of 1 lb. per ton of ore, then $1 + 2000$ or 0.0005 times the weight of ore, in lime, is added for each pound per ton required.

added to 13 gr. of ore for each pound of lime required.

The reaction of lime with oxalic acid is represented in an equation given above, from which it is seen that 126 parts, by weight, of oxalic acid reacts with 56 parts, by weight, of CaO. Therefore, 0.00146068 gr. of acid must be contained in each cubic centimeter of solution if one cubic centimeter is to represent a pound per ton of lime to be added, or 14.6068 gr. per liter.

If a decinormal solution is used: 1 c.c. taken in titrating 10 c.c. of solution represents the following amounts of alkalinity:

0.04 per cent NaOH, or 0.8 lb. per ton of solution.
0.028 per cent CaO, or 0.56 lb. per ton of solution.

It may be convenient in plant practice to adjust solutions so that 1 c.c. of the standard solution, when used on 10 c.c. of cyanide solution, will indicate a protective alkalinity or an acidity equal to 1 lb. CaO (unslackened lime) per ton of solution. For this purpose, 1000 c.c. of the standard solution must contain the following quantity of its appropriate reagent:

H₂SO₄, 8.75 gr. or 4.746 c.c. (sp. gr. 1.845).
HCl, 6.51 gr., or 5.42 c.c. (sp. gr. 1.20).
HNO₃, 11.25 gr., or 7.92 c.c. (sp. gr. 1.42).
Oxalic acid, H₂C₂O₄·2H₂O (solid), 11.25 gr.
NaOH (solid), 7.143 gr.
KOH (solid), 10.02 gr.
Na₂CO₃ (solid), 9.464 gr.

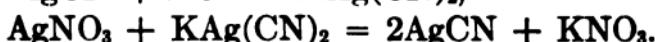
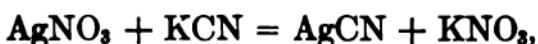
Titration for Cyanide Strength. — This is one of the commonest and most important tests, as

TITRATION FOR CYANIDE STRENGTH 105

it serves as a guide to the quantity of solid potassium and sodium cyanide that must be added to the solution. The result of this test indicates in terms of potassium all the cyanogen present in the solution as simple cyanides, that is, KCN, NaCN, etc. This test does not indicate all of the cyanide that has a solvent power on gold and silver, but it gives the strength obtained when making up new solutions, or that additional strength obtained by adding the solid cyanide to a solution already in good working condition.

The standard silver nitrate solution is made by dissolving 6.5232 gr. of pure crystallized silver nitrate in distilled water and diluting to 1000 c.c. Each cubic centimeter of this standard solution used in titrating 10 c.c. of cyanide solution represents one pound of KCN per ton of solution. The solution to be titrated should be perfectly clear, and the titration should be carried to the first faint turbidity. When titrating weak solutions (a fraction of a pound per ton), as much as 50 c.c. should be taken, the result being divided by 5. For a strong solution, after 10 c.c. has been measured out it is well to dilute to, say, 50 c.c., as the turbidity is then more easily seen.

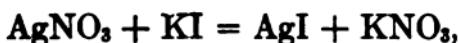
The reactions accomplished in the titration with silver nitrate are as follows:



On the first addition, the temporary white precipitate is due to the formation of AgCN;

this dissolves in the excess of cyanide to form the double cyanide $\text{KAg}(\text{CN})_2$. After all the cyanide has been converted to this compound the addition of a drop of AgNO_3 solution will cause a permanent white turbidity.

The titration of fresh solutions gives no difficulty, but with more complex mill solution the end point may be somewhat obscured and indefinite. In this case, the addition of a few drops of a 3 to 4-per cent solution of potassium iodide will obviate this chance of error, the titration being carried to a yellow color. The following reactions occur:



The final reaction produces silver iodide, when no more free cyanide is present, thus imparting a yellowish color to the solution.

Titration for Hydrocyanic Acid. — Hydrocyanic acid may be determined in cyanide solutions by first estimating the free cyanide, as above; then adding an excess of a solution of potassium or sodium bicarbonate, and titrating with the standard silver nitrate solution without the indicator. The difference between the result of this titration and that for free cyanide is taken as hydrocyanic acid.

The solution of potassium or sodium bicarbonate reacts with hydrocyanic acid as follows:



TITRATION FOR TOTAL CYANIDE 107

Since the HCN is titrated as KCN, the factor for reducing to HCN is

$$\frac{1 \ 12 \ 14}{\begin{matrix} H & C & N \\ K & C & N \end{matrix}} = \frac{27}{65.1} = 0.415.$$

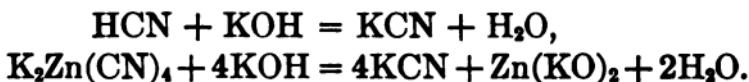
3.91 12 14

The result of the above titration, less that for free cyanide, indicates directly the pounds of potassium cyanide to which the cyanogen of the hydrocyanic acid is equivalent; or when multiplied by 0.415 the result indicates the amount of hydrocyanic acid.

Titration for Total Cyanide.— This test is for the purpose of determining, in terms of potassium cyanide, all of the cyanogen or CN radical present in the form of simple cyanides, as determined by the tests for free cyanide, such as that present as HCN, $K_2Zn(CN)_4$, and other easily decomposed double cyanides.

Take from 10 to 50 c.c. of cyanide solution, add a few drops of potassium iodide and then a few cubic centimeters of a normal sodium or potassium hydroxide solution. Titrate with the standard silver nitrate solution as described for free cyanide, but carry the titrations past a white turbidity until a permanent yellow color is obtained. The number of cubic centimeters of silver nitrate solution used for 10 c.c. of cyanide solution indicates, in terms of potassium cyanide, the number of pounds of total cyanide in a ton of solution. It should be observed whether increasing the amount of alkali added will increase the total cyanide obtained, the highest result being taken.

The reaction with potassium hydroxide in this test is represented by the following equations:



Test for Double Cyanides. — While this is an important test, it is made only occasionally by the chemist in charge. Its importance arises from the fact that it has been found, both in laboratory experiments and plant practice, that zinc-potassium cyanide is an active solvent for both gold and silver, and more active when apparently regenerated into KCN or a simple cyanide by the addition of alkali, as shown in the preceding equation. It will be seen that, knowing the amount of free and total cyanide, the quantity of double cyanides can be ascertained.

Ferrocyanides and Ferricyanides. — These may be determined by evaporation with acids and determination of iron. Ferrocyanide may be estimated by titration with KMnO_4 in the presence of free acid; N/100 KMnO_4 is preferable. Other methods are available (see Sutton). Ferricyanides rarely occur in ordinary working solutions. When oxidizing agents are absent they may be determined by liberation of iodine from potassic iodide, and the iodine titrated with N/10 thiosulphate and starch.

Alkaline Sulphides. — The presence of alkaline sulphides may be determined by agitating 200 c.c. of the solution with a small quantity of lead carbonate, PbCO_3 . A black precipitate of lead sulphide will indicate the presence of alkaline

sulphides. Another method requires the preparation of a solution of nitroprusside of potassium. Add a few drops of the nitroprusside solution to the cyanide solution; if alkaline sulphides are present, even in minute quantities, the solution will assume a brilliant purple color.

EXTRACTION TESTS.

Bottle Tests. — It is advisable to run a series of tests to determine each unknown factor, in each succeeding test using the best condition as determined in the preceding test. The following order is therefore recommended:

1. Best working strength of alkalinity.
2. Best strength of cyanide solution under best condition of alkalinity, as determined in test 1.
3. Most economical time of dissolution, using best conditions as determined by tests 1 and 2.
4. Relation of degree of fineness to time and percentage of extraction, using best conditions as determined by tests 1, 2, and 3.
5. Effect of variation in ratio of solution to ore, using best conditions as determined by tests 1, 2, 3, and 4.

Proper Strength of Alkali. — Take six fruit jars; in each put 100 gr. of 100-mesh ore and varying amounts of lime, say, from no lime in No. 1 to 10 lb. per ton in No. 6, No. 2 having 2 lb. per ton more than No. 1, and No. 3 having 2 lb. per ton more than No. 2, etc. Now add 100 c.c. of cyanide solution, having an arbitrarily chosen strength, say 3 lb. per ton, and shake well before placing on the agitator. Agitate for a given

length of time, say 24 hr. each. Remove bottles from agitator and allow to settle. Draw off 50 c.c. of the clear supernatant solution and titrate with standard silver nitrate, recording the number of c.c. required, which, divided by 5, gives the number of pounds of "free" cyanide per ton of solution present. The difference between this and the number of pounds added is recorded as "cyanide consumed" in pounds per ton. To the same solution now add 2 drops of phenolphthalein, and titrate with standard oxalic acid solution to the disappearance of the pink color. The number of $\frac{1}{10}$ c.c. required represents the points of alkalinity present.¹ The difference between this amount and that added equals the pounds of lime consumed.

The tailings from each sample should be treated as described under "Washing Residues" and assayed. The results of the above series is conveniently recorded in the form of Table III.

From the above table, a curve of extraction of gold and silver, one of consumption of KCN, and another of consumption of lime should be plotted, from which the variations, due to the varying amounts of lime added, in extraction and in KCN consumed, are plainly evident.

Best Strength of Cyanide. — In this test a series of six or more samples are run, using the alkalinity as determined above, but varying the

¹ One ton of ore = 2000 lb.; solubility of CaO in solution = 0.0013 lb.; therefore, amount of CaO, or 100 points of alkalinity, = 2.60 lb. CaO. One lb. CaO per ton of ore means 2.60 : 100 :: 1 : x ; x = 38.4 points.

TABLE III.

strength of cyanide, all other conditions being the same. At the end of the run the solutions are tested for free cyanide and protective alkalinity and the tailings are assayed for gold and silver. The results are recorded in a table similar to No. 3, from which curves of extraction, cyanide consumption, and consumption of CaO are made. From these, the economical strength, which must be balanced against cyanide and lime consumption, can be determined. It may be possible that a 6-lb. per ton solution will give a slightly higher extraction, but, on the other hand, with this strength of solution a higher consumption of cyanide will result, especially if bases are present, due to a less selective action for gold and silver. This must be taken into account.

Economical Time for Extraction. — Using the most suitable strength of cyanide solution and alkalinity, run a series of tests, varying the time of agitation, say 6, 14, 24, 32, and 48 hr., removing each at the expiration of the given time. Titrate for protective alkalinity and cyanide consumption, and assay tailing for percentage of extraction. Tabulate results, and plot curves of extraction and consumption of cyanide and lime against time of agitation. From these curves the time of agitation must be balanced against increased extraction and also against cyanide consumption. It may be more economical to take, for example, an extraction of 85 per cent in 14 hr. with a cyanide consumption of 1 lb. per ton than an extraction of 90 per cent requiring 24 hr. and consuming 2 lb. KCN per ton of ore.

Fineness of Ore. — Supposing it has been found from tests 1, 2, and 3, on 100-mesh ore, that the following conditions are best. Alkalinity, 5 lb. per ton; KCN, 3 lb. per ton; economical time of agitation, 14 hr. With these conditions, run a series of tests varying the fineness of the ore, say 60, 80, 100, 150, and 200 mesh. At the end of 14 hr. remove all samples, and make the usual tests. Plot extraction and cyanide consumption against degree of fineness. From this, the fineness required for maximum extraction can be seen. Suppose it is found to occur with the 200-mesh material. With ore of this size another series should be made, again varying the length of time, for it is possible that maximum extraction on such fine material may be reached in 2 hours agitation. Then this reduction in time should be balanced against increased cost of grinding and diminished capacity of the plant.

Ratio of Solution to Ore. — It is well to run tests, varying the ratio of solution to ore, say 0.75 : 1, 1.25 : 1, 2 : 1, 2.5 : 1, and 3 : 1. The circulation of a large amount of solution per ton of ore may give increased extraction.

From the above series of bottle tests, some idea as to the refractoriness of the ore will be obtained. The question now lies as to whether an increased extraction can be secured by any other method. The samples having been agitated in the absence of free air, the logical test is to see whether agitation with air, using conditions as obtained in the bottle tests, will yield any increase in extraction.

Air Agitation Tests. — These should be made on 150-mesh material, to correspond with practice, as carried out in the Pachuca tank. Various laboratory devices have been used for making these tests. A suitable one consists of a large funnel, 10 in. diameter, set up as shown in Fig. 12. The rubber tube should be looped up to a level above that of the pulp in the funnel so that if the air pressure accidentally stops the pulp will not make its escape. Another convenient device is shown in Fig. 13. It consists of an Erlenmeyer flask, of 1000 c.c. capacity or larger, fitted with glass tubing and suction as indicated; the application of a vacuum causes air to be drawn in through *b*, passing up through the siphon and

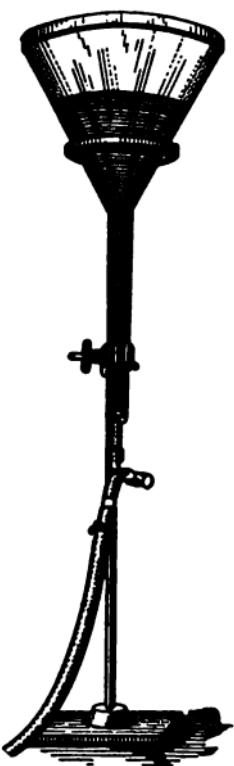


FIG. 12. Laboratory Agitator.

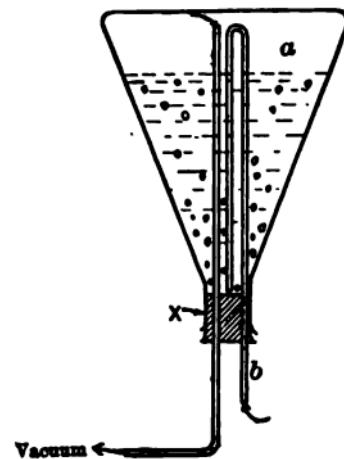


FIG. 13. Laboratory Flask Agitator.

sure accidentally stops the pulp will not make its escape. Another convenient device is shown in Fig. 13. It consists of an Erlenmeyer flask, of 1000 c.c. capacity or larger, fitted with glass tubing and suction as indicated; the application of a vacuum causes air to be drawn in through *b*, passing up through the siphon and

impinging against the cork *x*. The bend in the tube in the vacuum chamber *a* prevents pulp from escaping from the agitator, and the suction may be shut off without fear of losing pulp. Four or five of these can be operated from one common suction flask attached to an aspirator.

Ball-mill Tests. — These can be made on quantities ranging from one to several pounds of ore, and are conducted in cyanide solution, tending to duplicate "all sliming" processes as used in mill practice. Ordinarily a high extraction can be obtained in this manner, due to the agitation, the extremely fine size of the particles, and the brightening and polishing effect on the metals. With many ores even this treatment will fail to give a high extraction, especially with telluride, antimony and barytc ores. An extra chemical must usually be resorted to for trial.

A suitable ball mill for these tests is shown in Fig. 14. Place the sample of ore (crushed to 10 mesh) in the mill, with the desired amount of lime or other alkali, and the required amount of cyanide solution; the ratio of solution to ore should be approximately 1 to 1. Clamp down the lid and place the jar in its frame for rotation. The speed should be carefully adjusted to obtain a maximum crushing effect; it is easy to judge by the ear when the pebbles are dropping properly. If it revolves too slowly the crushing is slow; if too rapidly, centrifugal force causes the balls to be thrown against the periphery of the jar, and if fast enough, crushing may cease entirely.

Usually from one-half to three hours is sufficient duration for a ball-mill test. At the end of each test the solution is to be titrated for protective alkalinity and free cyanide, and the tailings assayed to determine the percentage of extrac-

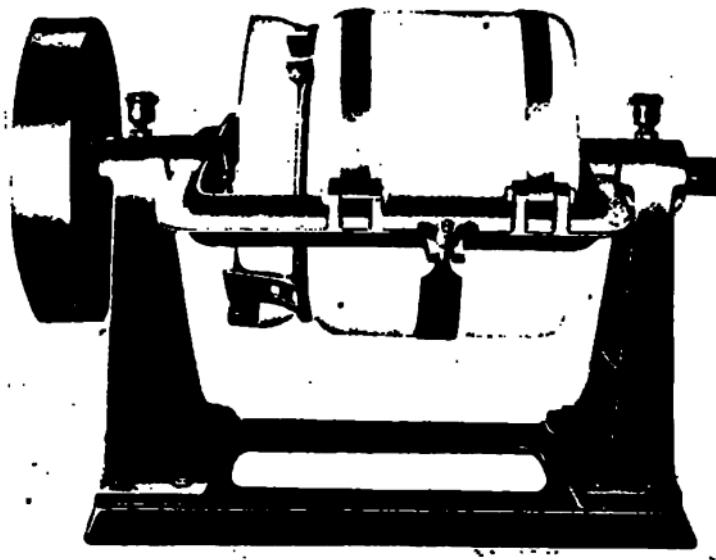


FIG. 14. Laboratory Ball-mill.

tion. The results should then be compared with those obtained by bottle tests to see if an increased extraction has been obtained. If an ore refuses to give up an appreciable amount of its value by fine grinding in cyanide solution, as thus conducted, it is probable that an extra chemical must be resorted to, or else the physical and chemical nature of the ore must be changed by roasting.

Tests with Supersolvents. — These chemicals are resorted to only when ordinary methods fail to give a good extraction, for the reason that

their use, especially bromocyanide, requires extra care and added expense. Mercurous, Hg_2Cl_2 , or mercuric chloride, $HgCl_2$, have been used and in some cases give an increased extraction. (For the part played by supersolvents, see under "Supersolvents," Chapter IV.)

Roasting Preliminary to Cyanidation. — The objects of roasting an ore before subjecting it to the solvent action of some chemical, are:

1. To oxidize iron, copper, lead, zinc, aluminum, calcium, or magnesium.

2. To volatilize sulphur, arsenic, antimony, bismuth, or tellurium.

3. To sulphatize calcium, magnesium, and to some extent, lead, and zinc.

4. To dehydrate, with the object of agglomerating the ore particles to make them more susceptible to leaching, decantation, or filtration. Usually an ore, which will percolate or filter very slowly before roasting will percolate or filter quite rapidly after roasting. Mill dust, when raw, may be difficult to filter, but after roasting filtration is quite rapid.

5. To make the ore porous. Oxidation of sulphur and telluride ores, by the elimination of sulphur and tellurium, must necessarily make the ore particles more porous, and thus present a greater surface to the action of the solvent. Most ores will yield a much higher extraction after roasting than before, even though they are otherwise equally susceptible to treatment.

6. To free gold and silver particles. Cyanide is not a practical solvent of gold and silver in the

telluride combination, and in their sulphide condition they present serious difficulties. After roasting, the gold and silver are reduced to their metallic state and are then readily soluble if the particles have not been fused.

Since the cost of roasting entails a heavy additional expense, the increased extraction resulting from the roasting of the ore must be carefully balanced against the extra cost of roasting. After roasting, an ore may be so leachable as to make fine grinding unnecessary, thus giving a small margin of expense to be deducted from the cost of roasting. Tests on roasted ore are conducted in the same manner as on raw ore; perhaps a water wash before cyaniding will be of value.

Percolation Tests. — The term "percolation" is synonymous with "leaching;" however, the former refers more particularly to the passing of a solution through a porous medium, while the latter relates to the dissolving and removing of some substance from the material through which the solution is passing.

In making laboratory tests, the main object should be, as nearly as possible, to duplicate the conditions of actual practice. This point is, however, often overlooked, and in my opinion a great majority of the tests made to determine possible extractions by leaching are of little value. Usually, the absurdly small amount of sand taken for treatment in preliminary experimental work leads to the obtaining of unreliable data, as the depth of material is too shallow to demonstrate the

chemical changes in the solution, and the resultant effects occurring in an ordinary leaching vat.

Fig. 15 shows a laboratory apparatus for leaching experiments, by G. A. James.¹ It is claimed

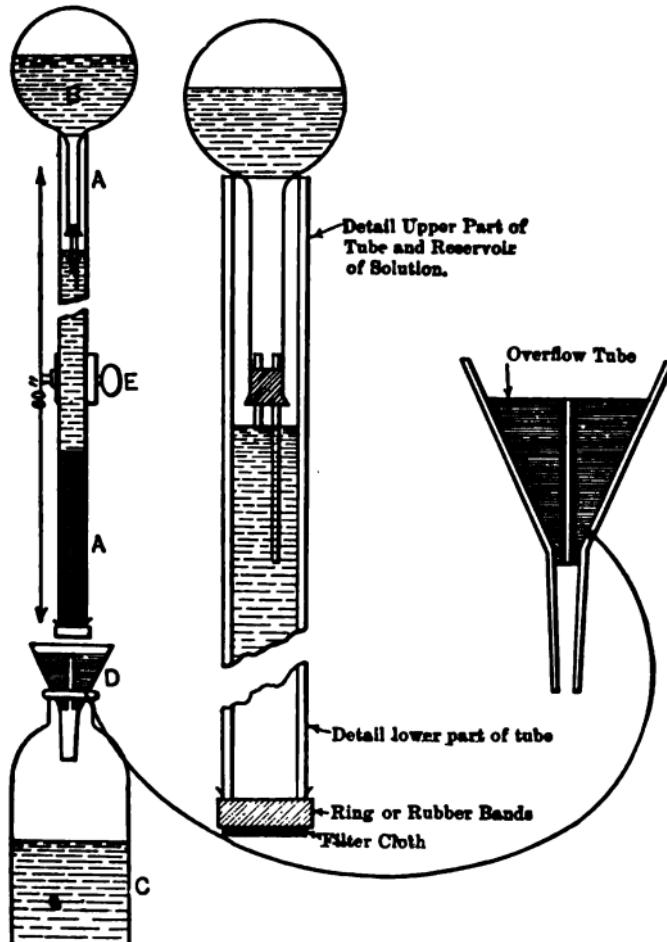


FIG. 15. James' Laboratory Leaching Apparatus.

by Mr. James that this apparatus gives results corresponding closely to those obtainable in practice. It consists, as shown, of a large glass

¹ *Min. & Sci. Press*, Dec. 30, 1911.

tube, *A*, 60 in. long and not less than 1.25 in. internal diameter (1.5 in. is preferable). The lower end is covered with a filter medium of cloth or canvas, which is held in place by a tight fitting ring of metal or by rubber bands. The 2000-c.c. measuring flask, *B*, should have a neck small enough to enter the glass tube. This is fitted with a cork, through which two small tubes enter the neck of the flask. One protrudes about an inch and the other must be at least three inches longer. The purpose of this arrangement is to permit a self-regulating feed to the percolation tube, *A*, and to prevent the contamination of the solution by the laboratory gases. As the shorter tube provides the only inlet for air, its length regulates the height of the liquid column in *A*. *D* is a funnel with an overflow tube covered with a small piece of rubber tubing, which may be pressed into the neck of the funnel, thus retaining sufficient solution for testing. This always represents the last liquid received, and can be allowed to enter the solution reservoir by releasing from the neck of the funnel.

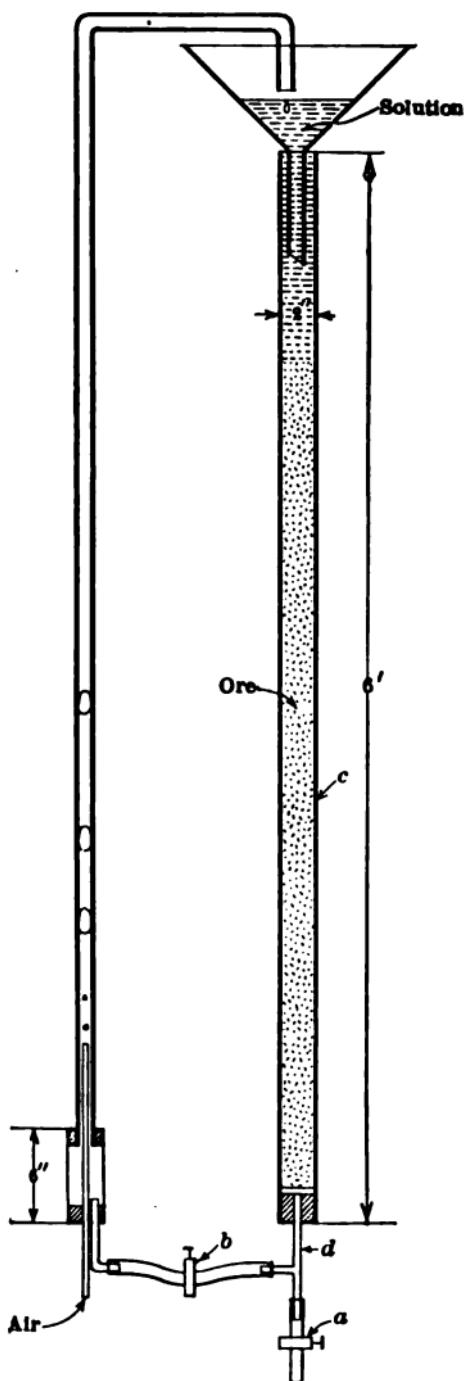
The above method is open to criticism. In the leaching of silver-gold ores the relation of depth of sand to percentage of extraction is an important question; in this apparatus only three feet of ore is leached. A leaching experiment should be so arranged that the bottom and top of the charge may be sampled at intervals. No arrangement is provided for intermittent application of solutions or solution washes, which is advantageous where reducing action is evident.

A third objection is that no contrivance is provided by which the affluent can be shut off, or by which the rate of leaching can be controlled. The preliminary soaking of the ore in strong solution necessitates some valve arrangement, and if this method of uncontrolled gravity leaching were adopted in practice with a coarse clean sand the pumping expense would be prohibitive. The determination of *maximum* percolation rate is of secondary importance to the determination of *efficient* percolation rate, since the rushing of solution through a bed of sand requires the handling and precipitation of large quantities of low-grade solution, and entails the production of base precipitate, with its attendant expense and trouble.

A. W. Allen¹ recommends the use of a 4-in. pipe in the experimental determination of cyanide extraction by leaching. The pipe is fitted with bored flanges, at one end for suspension and at the other for the filter bottom. The latter is kept in place by a third flange, with insertion joint, and fitted with a plug tapped for a small leaching control pipe, an arrangement incidentally providing for the upward charging of solution. The pipes should be cut or compounded of a length equal to the height of the vat it is proposed to use in practice.

Another convenient apparatus is that shown in Fig. 16, which I have used with satisfactory results for some time. This apparatus is continuous, the solution being caused to circulate

¹ *Min. & Sci. Press*, Feb. 17, 1912.



FIG

hor's Automatic Leaching Apparatus.

by an air-lift arrangement, as shown. A tap plug *a* gives a means of securing solution for test. Valve *b* controls the rate of leaching. This causes complete aeration of the solution and at times the ore may be completely drained, where reducing action is evident. The leaching rate is the fall of the surface of solution in inches per hour when the charge is completely covered.

CHAPTER IX.

LEAD SALTS IN CYANIDE PRACTICE.

Lead salts are used in cyanidation for the purpose of either increasing the extractive power of the solution or assisting precipitation. The addition of a lead salt for either one of these specific purposes does not necessarily mean that the lead will be present to serve in the other capacity; for, when only the proper proportion is used to give maximum extraction, there may be little or no lead in the solution when it reaches the precipitation stage; and, when used to aid precipitation, metallic lead is precipitated by the zinc, and, therefore, remains with the precipitate.

Lead Salts and Extraction. — It has been shown definitely by Clennell,¹ Hamilton,² Clevenger,³ and others,⁴ that soluble sulphides do not

¹ "Soluble sulphides are not stable in cyanide solutions and can seldom, if ever, be detected," "Lead Salts in Cyanide Treatment," J. R. Clennell, *Eng. & Min. Jour.*, Vol. 94, p. 597.

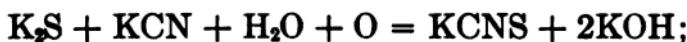
² "I have never succeeded in finding soluble sulphides in any solution used for treating a silver ore, although sulphocyanate is almost invariably present," "Lead Acetate in the Cyanidation of Silver Ores," E. M. Hamilton, *Mex. Min. Jour.*, Vol. 11, p. 47.

³ "I have very rarely detected soluble sulphides in the mill solution." Private reports from G. H. Clevenger.

⁴ Anonymous discussion of "Lead Salts in Cyanide Treatment," by J. E. Clennell, *Eng. & Min. Jour.*, Vol. 94, p. 921.

exist in mill solutions, especially when zinc precipitation is employed, at the time that they come in contact with a fresh charge of ore. There are, however, those who adhere to the idea that the principal function of lead salts in cyanidation is to remove soluble sulphides from solution. While it is true, that lead salts will do this, since it is only in exceptional cases that solutions contain soluble sulphides, the futility of this procedure is apparent.

There are three reactions which prevent the formation of soluble sulphides: (1) Zinc taken into solution during precipitation appears to be as effective for their removal as lead salts added intentionally; (2) Sulphides rapidly form sulphocyanates with the simple alkaline cyanides which are always present, as:



(3) Alkaline sulphides rapidly oxidize to thiosulphates, which in turn probably largely oxidize to sulphates; these cannot rise above a small percentage on account of the slight solubility of calcium sulphate.

The following are conclusions drawn by Clevenger¹ regarding lead salts.

(1) Lead salts may serve to keep down the concentration of soluble sulphides in solution, but they seldom operate in this capacity for the reason that soluble sulphides are rarely present in mill solutions.

¹ "The Function of Lead Salts in Cyanidation," *Min. & Sci. Press*, Oct. 24, 1914.

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(2) They may accelerate the dissolution of silver from ores in which silver occurs as sulphide, through the more prompt removal of the sulphur ion from the system.

(3) They may, in certain cases with sulphide ores, effect a saving of cyanide through retarding the formation of sulphocyanate or preventing the separation of elemental sulphur from the solution.

(4) They may be a benefit when a portion of the silver is intimately associated with copper; also when the concentration of copper in the solution rises to a dangerous point, they may mitigate its fouling tendency. However, this statement does not mean that the use of lead salts is a universal panacea for all the ills due to copper cyanidation. So far as we know, it is only where the effect is not too great that this remedy is useful.

(5) A great variety of the compounds of lead may be used for this purpose. The compound used should depend upon the cost of its lead content as well as its convenience in use. The compound most generally used in the past has been the acetate, although litharge has been claimed equally satisfactory. The cost of the lead content of litharge is perhaps lower than any of the other compounds, but it is more difficult to apply, as it is necessary to grind it very fine and exercise care to see that it is thoroughly mixed with the charges undergoing treatment. However, an excess of litharge appears to be less detrimental than an excess of a soluble salt.

(6) Too great care cannot be exercised in adding

the proper proportion of lead salt. This can be determined only by careful experimentation in each particular case, and should be checked at frequent intervals.

(7) A number of those who have written on the subject are of the opinion that lead salts may be of advantage in starting a new plant with fresh solution, but that this advantage disappears after the solutions have been in use for some time. This clearly points to the fact that there are a number of substitutes which may perform the same functions as the lead salt. Perhaps chief among these is zinc taken into solution during precipitation. On the other hand, my own experience in at least one case, when treating a silver-gold ore, was that the use of a lead salt was more advantageous with the mill solution than with fresh solution. Although there might be other factors which might enter in, the presence of copper in solution possibly explains this contradictory behavior. Small amounts of mercury escaping from amalgamation processes preceding cyanide treatment would operate more or less in the same way as lead salts. Aération might also, at times, be an efficient substitute.

(8) On the whole, the advantages arising from the use of a lead salt are more apparent when treating silver ores, or silver-gold ores in which silver occurs in combination with sulphur, than when treating gold ores. In ores in which silver is combined with antimony it is likely to be a detriment. It may be said that the advantageous use of lead salts is one of the finer points of the

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operation, and, even where its use might be advantageous, unskilled application may spoil the whole effect.

Lead Salts and Precipitation. — Lead salts may aid precipitation by forming a zinc-lead couple when zinc is used as a precipitant; this is especially true when precipitating from weak cyanide solutions.

Julian and Smart¹ state that the general method of preparing the zinc-lead couple is to immerse the zinc shavings in a 1 to 5-per cent solution of lead acetate prior to packing them in the zinc-boxes. Carter² calls attention to the importance of a proper preliminary preparation of the zinc for this method of precipitation. He suggests stirring the freshly cut zinc shavings in a 10-per cent solution of lead acetate. If the shavings are not thoroughly stirred in the solution, only the outside of the mass becomes properly coated with lead. After preparation, the shavings should be placed in the precipitation box at once, as exposure to air results in rapid oxidation of the zinc.

In use, the lead-zinc couple gradually becomes less and less active until a point is reached when it no longer properly performs the function of precipitation. This makes necessary more frequent clean-ups, and involves the destruction of a greater proportion of zinc by acid than when the lead couple is not used.

¹ "Cyaniding Gold and Silver Ores," H. Forbes Julian and Edgar Smart, London, 1907, p. 138.

² Loc. cit., p. 441.

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The following résumé¹ gives the functions of lead salts in precipitation:

(1) The zinc-lead couple is of particular value in precipitating solutions containing copper, or gold solutions containing a low percentage of cyanide.

(2) In the majority of cases, where the solution contains an appreciable amount of silver, the lead couple is unnecessary, although the presence of a small percentage (2 to 3 per cent) of lead in the zinc dust, if this method of precipitation is used, is generally favored by most operators. Most of the commercial sheet zinc from which zinc shavings are cut contains upward of 1 per cent of lead, so that, to a greater or less extent, the zinc-lead couple is involved in all systems of zinc precipitation.

(3) For forming the zinc-lead couple, it is necessary to use a soluble lead salt. Most frequently the acetate has been used for this purpose on account of being the most readily available commercial salt, but the nitrate has been used at times on account of its higher lead content.

(4) The zinc-lead couple necessitates the frequent removal of the inactive precipitant, which is a decided disadvantage, even when used for precipitating stronger solutions, for inevitably some oxidation takes place in handling the decomposed zinc. On account of the constant addition of fresh precipitant and the comparatively small

¹ "The Function of Lead Salts in Cyanidation," by G. H. Clevenger, *Min. & Sci. Press*, Oct. 24, 1917.

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excess of precipitant remaining in the precipitation press, zinc-dust precipitation seems to lend itself more readily to precipitating coppery solutions, or for the application, in general, of the zinc-lead couple.

CHAPTER X.

RECOVERY OF GOLD AND SILVER SOLUTIONS FROM PULP.

Two systems are employed for separating clear gold or silver-bearing solutions from the ore pulp: (1) Decantation; (2) filtration.

DECANTATION.

Intermittent Decantation. — This process which was the earliest to be applied, consists in mixing with the pulp containing the values in solution a solution of lower-grade contents, settling the mixture in a tank, and decanting the clear supernatant fluid. The thick pulp remaining in the tank is pumped to a second tank, together with more barren solution, and again settled and decanted. After several repetitions of this operation, the values are so far reduced that further washing is not profitable. The gold recovery by this process is high, but the plant required is bulky, labor cost is high, and the amount of solution to be precipitated is excessive.

Continuous Decantation. — As early as 1901, a plant was built in the Black Hills, of South Dakota, by John Randell, employing the former principles but attempting to make the plant continuous; for flat-bottomed tanks, he sub-

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stituted cones which operated continuously, receiving a constant feed and discharging a steady stream of thickened pulp. These cones were operated in series, the thickened underflow of the first one forming, with a stream of diluting solution, the feed to the second cone of the series. Barren solution was added to the tank immediately preceding the discharge tank, and after being slightly enriched by the low-grade pulp in this tank, overflowed to yield a diluting solution for the richer feed entering the third tank. This is the principle upon which all successful "countercurrent decantation" plants operate at the present time, but Randell's plant was not successful because of mechanical difficulties in getting a continuous thick discharge from his cone tanks. A similar plant was built in South Africa, although there the washes were not used repeatedly, as in Randell's system, but were precipitated after each contact with the ore. This also was abandoned on account of mechanical difficulties and the cost of precipitating the large quantities of solution that had to be used. For a number of years the process was not used, and it was not until the introduction of the Dorr thickener that the minds of the metallurgists turned again to the continuous decantation principle.

In 1901, two decantation plants were built, making use of flow sheets similar to that used by Randell nine years before, but substituting Dorr thickeners for cones. Several plants, working on this system, are now in operation.

Countercurrent Decantation. — Fig. 17 represents a typical flow sheet of a countercurrent system. This assumes that crushing is done in solution, the overflow from tank T_2 being used for that purpose. On leaving the grinding circuit with the ground pulp, the solution enters T_1 , and that part which does not pass with the pulp to the agitators, A , overflows T_1 and goes to precipitation. After depositing its gold con-

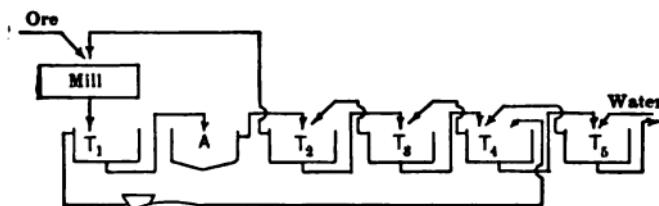


FIG. 17. Typical Countercurrent System of Decantation.

tents, it is used to dilute the underflow of T_3 as this enters T_4 . The overflow of T_4 mixes with the underflow of T_2 to provide the feed to T_3 , etc., as indicated in the flow sheet. As each succeeding mixture meets pulp richer in dissolved metals than itself, it is enriched while the pulp is correspondingly impoverished. The pulp at each step approaches the discharge end of the mill while the solution goes towards the feed end, hence, "countercurrent decantation."

Variables Affecting the Decantation Process. — The principal variables that may affect the efficiency of the decantation process are:

1. Grade of ore.
2. Ratio of solution precipitated to ore treated.
3. Thickness at which pulp can be discharged.
4. Cost of chemicals.

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5. Rapidity of dissolving and the place in the circuit where it takes place.

6. Efficiency of precipitation.

For a complete theoretical discussion of this subject the reader is referred to an article by Luther B. Eames, "Countercurrent Decantation," *Bul. A.I.M.E.*, Dec., 1916, p. 2087, in which the above variables are mathematically studied.

FILTRATION.

Classification of Machines. — The mechanical appliances in use for filtration are grouped as follows¹:

1. Suction filters, in which a vacuum is applied to accelerate filtration.

A. Appliances making a thin slime cake, and practically continuous in their operation (Oliver and Ridgway.)

B. Appliances making a thick slime cake, and intermittent in their operation. (Moore and Butters.)

2. Pressure filters, in which hydrostatic head, compressed air, or pumps are used to secure greater pressures than are possible with a vacuum pump. These filters are intermittent in their action.

C. Ordinary filter presses.

D. Sluicing filter presses. (Merrill.)

E. Filtering chambers or cylinders, in which the filtering basket is inclosed. (Burt, Kelly, and Sweetland.)

¹ "Slime Filtration," by Geo. J. Young, *Trans., A.I.M.E.*, Vol. XLII.

3. Centrifugal filters, in which centrifugal force is applied to separate solution from slime. These filters are continuous in action.

Advantages of Continuous Filters.—The effort to secure a continuously acting filter has resulted in the development of two important types, of which the Ridgway and the Oliver are the best known. Both of these filters make a comparatively thin slime cake; both operate successfully, and compared with the thick-cake machines have decided advantages, briefly stated as: Simplicity of design; probably lower capitalization charges for equal capacities; lower operating costs; and less attention required in their operation.

Operation of Filters.—With the exception of the Oliver filter, the general method of operating both suction and pressure filters is the same. The slime pulp is delivered to the filter in the proportion of one of dry slime to from three to one of solution. The pulp is forced into the cells of the pressure filters and a cake is formed against the canvas walls; the surplus pulp, if any, is withdrawn, and wash water forced in until the contained solutions are displaced. The cake is then forced off from the canvas surface, either by water or by compressed air, or both, and sluiced out. In the vacuum filter the filtering cells are immersed in the pulp, a vacuum is formed, and a cake built up; the surplus pulp is then withdrawn either by lifting the filtering cells or by withdrawing the pulp with pumps, and the cakes are immersed in water for washing. In

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the Moore filter the cakes are discharged by forcing them off from the cells by water or air and dropping them into a hopper for sluicing away. In the Butters filter, the cake is forced off in the same way, but while still immersed in the wash solution. The wash solution is then withdrawn

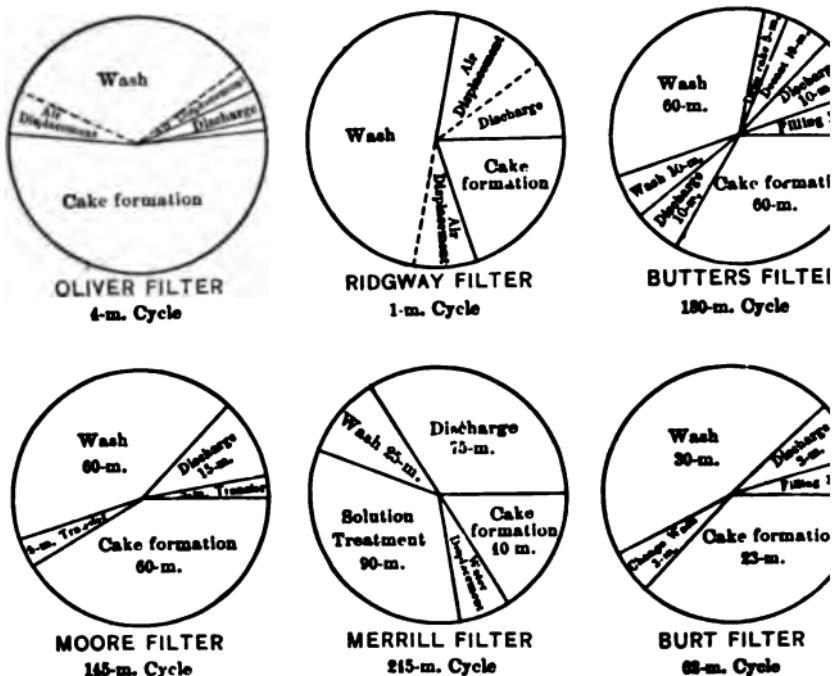


FIG. 18. Slime Filtering Cycles.

either by decanting or pumping, and the slime cake and surplus water sluiced out. The Oliver filter performs the operation of cake formation, washing, and discharge in continuous sequence. Three steps are common to all of these filters: cake forming, washing, and discharge. The ~~order~~ of operations of the more common filters

is shown in Fig. 18, typical examples having been taken in every case.¹

Filter Medium and Support. — Most of the suction filters employ No. 10 duck for filtering surface. The Oliver filter uses No. 12, and the Merrill filter press uses No. 6 duck over a light twill. In the Butters and the Moore filters, three methods of support are in common use. The original Butters unit consisted of canvas stitched at close intervals over a sheet of cocoa matting, which gives a very porous gathering space for the solution and also sufficient support for the canvas. The objections to this construction are the cost and the clogging of the matting. With the exception of the Goldfield Consolidated mill, all the mills in the Tonopah and Goldfield districts employ the "slat method" of support, which consists of sewing the canvas walls of the cell into narrow pockets from 1 to 1.5 in. wide, and into each of these slipping a grooved lath. The arrangement is low in first cost and very satisfactory. The Moore system employs wooden strips slipped into narrow pockets in the canvas; and also uses wire netting between the canvas and walls, the canvas being stitched at frequent intervals through the netting. In the Oliver filter, wire netting over a grooved board, and covered with 8-oz. burlap, supports the canvas; the canvas is held against this base by wire wrapped around the canvas at 0.5-in. intervals. In both the Butters and the Moore filters, wooden

¹ "Slime Filtration," by George J. Young, *Trans., A.I.M.E.*, Vol. XLII.

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dividing strips are used to space the filtering surface into strips 1 ft. wide. Grooved iron plates are used in the filter presses and in the Merrill press.

Durability and permeability are the necessary requirements of a filtering cloth. No. 10 canvas duck, army weave, answers both of these requirements for suction filters. For pressure filters this canvas is too light, and No. 6 gives sufficient durability without retarding the filtration too much. Fig. 19 illustrates several

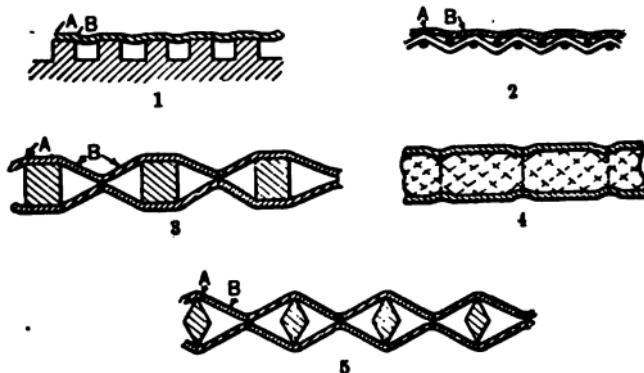


FIG. 19. Methods of Supporting Filter Cloth.

methods of supporting the filtering cloth. It can be seen that the relatively small proportion of ridge to groove determines the decrease in permeability due to support. In No. 5, the diamond shaped strips give the maximum proportion of distended canvas, and leave the fiber free from any flattening due to pressure.

Important Factors in Filtration.¹—

1. The proportion of clayey material in ores which are to be subjected to "all sliming" and filtration should be maintained at a minimum.

ious reference.

2. The slime pulp should be as free as possible from sands coarser than 150-mesh, and as large a proportion of the pulp as possible should consist of material passing 200-mesh.
3. The slime pulp, before filtration, should be

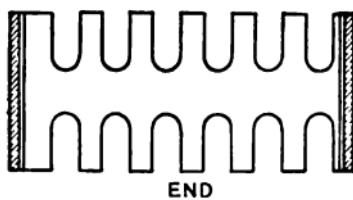
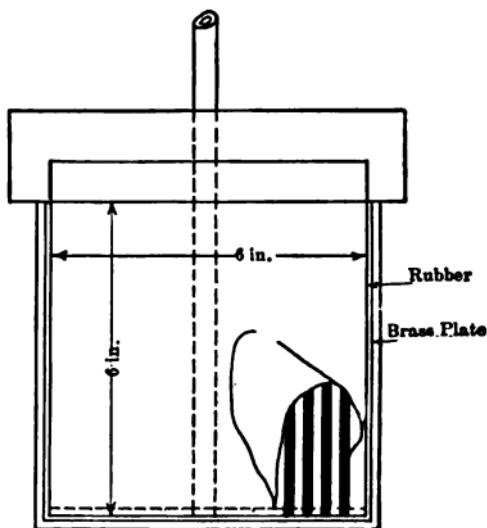


FIG. 20. Experimental Filter Leaf.

settled to as thick a consistency as can be handled by pumps and pipes.

4. The temperature of the wash water should be the same as that of the pulp.
5. The temperature should be maintained between 20° and 30° C., or higher.

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6. Vacuum pressures should be varied until the proper intensity for the given slime is obtained.

7. Where very clayey slime is to be treated, as much fine sand should be crowded into the pulp as it will carry without undue settling and clogging.

8. No. 10 canvas supported by slats gives the

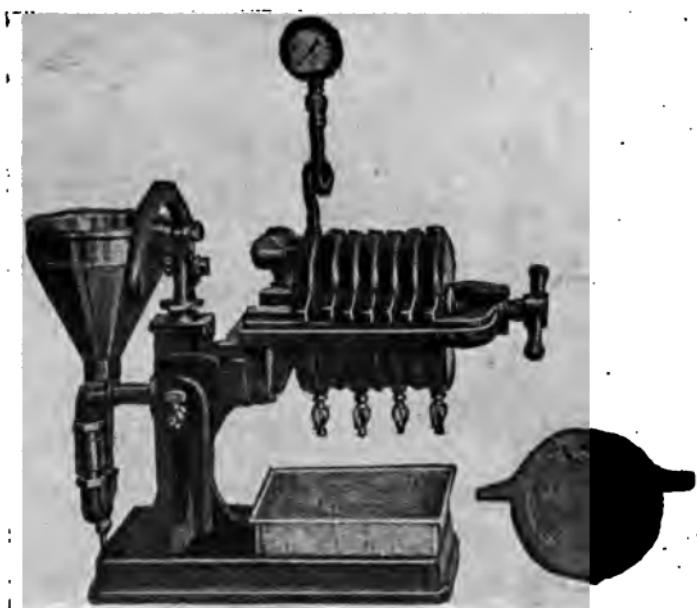


FIG. 21. Laboratory Pressure Filter.

best general service for thick-cake filters, and No. 12 canvas, on wire netting, answers the requirements for thin-cake machines.

9. With slimes containing a large proportion of colloid or clayey material, pressures greater than those obtainable with vacuum filter should be used.

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10. With slimes containing a small proportion of clayey material and much fine sand, both vacuum filters and pressure filters can be used, with perhaps equally good results.
11. With slime containing much coarse and fine sands, the chamber filters, with air-agitation and high pressures, would perhaps give the best results.
12. Of the vacuum filters, the thin-cake continuous machines are a decided improvement over the thick-cake filters.

Filter for Experimental Work. — Fig. 20 shows a small filter leaf that is probably suitable for vacuum filtration work. Small plate and frame filter presses, Fig. 21, can be obtained from any of the well-known laboratory furnishing houses.

CHAPTER XI.

PRECIPITATION OF GOLD AND SILVER FROM CYANIDE SOLUTIONS.

A cyanide solution of gold and silver, generally called "gold" solution or "pregnant" solution, cannot be tested in the laboratory with any certainty of obtaining results corresponding to those which would be obtained in practice. However, some idea can be obtained.

Classification. — The methods of precipitating gold and silver from cyanide solution may be classified as follows:

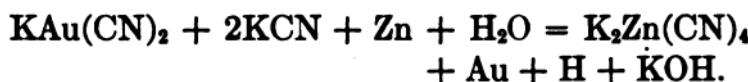
1. Precipitation with zinc.
 - a. As shavings.
 - b. As balls.
 - c. As dust.
2. Precipitation with aluminum.
3. Precipitation with sodium sulphide.
4. Precipitation by electrolytic methods.
5. Precipitation with charcoal.

PRECIPITATION WITH ZINC.

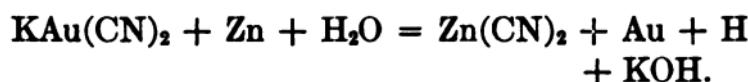
As Shavings. — During the early development of the cyanide process and until recent years, zinc shavings were used almost universally as a precipitant for gold and silver, and are still used to a large extent.

Reactions. — The precipitation of the precious metals from cyanide solution by zinc is due to the replacement of the gold and silver in the double cyanide by the zinc, and to electrolytic currents, produced by the chemical reactions, which electrochemically deposit the precious metals from the solution.

The precipitation of gold, and similarly of silver, in the presence of free cyanide may be expressed by the equation:



In the absence of free cyanide, as:



If no precious metal is present in solution, the following reaction will take place:



For formation and prevention of "white precipitate" see "White Precipitate," Chapter VI.

Zinc-box Troubles. — Troubles in the zinc-boxes, of which one used to hear and see so much, were usually one or more of the following:

- (1) White precipitate on the zinc.
- (2) Gelatinous silica forming in the boxes.
- (3) Hydrated alumina coming out in the boxes.
- (4) Coffee-colored solutions, which it was almost impossible to test owing to their color.
- (5) Muddy or slimy solutions.
- (6) Foul solutions.

The cause of any zinc-box trouble is usually

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to be found in one or more of the following conditions: Solutions entering the zinc-box without free caustic alkali, or even containing free acid; soluble magnesium salts in the ore or in the water used; copper in solutions; presence of alkaline carbonates or sulphides in the cyanide, allowing the hydrogen, etc., generated in the zinc-boxes, to accumulate instead of insuring its liberation about as fast as formed; greasy or oily zinc shavings; turbid solution, due to damaged filter cloths or to imperfect decantation, or to the presence of very small quantities of flocculent colloidal matter.

Function of Alkalies. — Solutions which do not contain free caustic alkali (lime), or do contain free acid, may carry in solution silica, magnesium, and aluminum salts, iron compounds, and organic matter. These substances can be carried only in acid or neutral solutions, and will be deposited on the zinc shavings, or partly as a surface scum in the zinc-box, as soon as the solution, during the process of working, becomes alkaline. As a result, the zinc shavings will be more or less coated and will gradually cease work. Organic matter, usually derived from ore which has been on the surface for a long time, gives the solutions a deep-brown color; but all this organic matter can be precipitated by caustic lime, leaving the solutions clear. If every solution, before it left the ore, were alkaline with caustic lime, all the substances referred to would have been precipitated right in the ore, and would remain there in their proper place.

Effect of Copper.—Copper in solution has a tendency to "plate" the smooth side of a zinc shaving, and when it has done so, no further precipitation will take place. Lead acetate and other means are employed to roughen the surface of the zinc, so that the copper, etc., may deposit as a slime and not as a "plating deposit."

Action of Carbonates.—When cyanide containing carbonates or sulphides is added to solutions entering the zinc-box, either in the solid state or by dripping in a strong solution, since the solutions contain both lime and zinc compounds, calcium carbonate and zinc sulphide are formed by decomposition. The former crystallizes on the zinc shavings and is easily detected by its roughness to the touch; the presence of this carbonate of lime will subsequently increase clean-up tests. The zinc sulphide forms a flocculent precipitate which settles on the zinc. The remedy is to be found in purchasing pure cyanide.

Test for Carbonates in Cyanide.—The following is a convenient test for carbonates:¹ Add 10 c.c. of a saturated solution of $\text{Ba}(\text{NO}_3)_2$ to a 25-c.c. sample of the solution; after filtering and washing the precipitate, the moist filter paper is placed in an Erlenmeyer flask. Add 10 c.c. of N/10 NaOH_3 and macerate the filter paper; titrate with N/10 NaOH , using 2 drops of methyl orange as an indicator. This method gives the carbonate, together with one-half of the bicarbonate.

¹ "Atmospheric Decomposition of Cyanide Solution," by G. H. Clevenger and Harry Morgan, *Min. & Sci. Press*, Sept. 16, 1916.

The sum total of all zinc precipitation is that the zinc works satisfactorily so long as the zinc in the precipitating arrangement, whatever it may be, is metallic zinc, and that the actual clean metal surface is directly exposed to the circulating cyanide solution.

Defects of Zinc-shaving Precipitation. — In the use of zinc shavings as a precipitant, many



FIG. 22. Standard Steel Zinc-box.

defects are exhibited, of which the following are the principal ones: The amount of zinc actually consumed is enormously above the amount theoretically demanded for each ounce of gold or silver; none of this excess zinc is recovered; the amount of handling of the zinc and gold slimes and the possibilities of loss are therefore increased; there is a loss of interest on the gold and silver values held in suspension in the zinc-boxes; the frequent cleaning-up and all the unsatisfactory handling are due to the fact that

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there is no means of removing, from the zinc, mechanically or otherwise, all the precipitates of whatever kind as fast as they are formed, thereby leaving the zinc surface always clean for further action.

Fig. 22 shows an ordinary form of zinc-box.

Precipitation with Zinc Balls. — A new method for precipitating gold and silver by zinc has been devised by H. R. Conklin;¹ its main purpose is to eliminate the use of zinc in the form of dust or shavings. It also has two other objects: first to apply the zinc precipitant in its cheapest form; and second, to obtain a precipitate free from impurities, especially from zinc.

The basic idea is to pass the pregnant solution over and through a quantity of small zinc balls, with other small pieces of zinc, the flow of the solution and a rolling action combining to remove the precipitate from the zinc, the final separation of the precipitate from the barren solution taking place after leaving the machine.

Pregnant solution enters the machine at the top, Fig. 23, flows down through the slowly turning zinc, follows the spiral path around the inner tube, and leaves the machine through an outlet pipe so designed as to prevent any zinc from passing out with the solution, which carries the precipitate with it. The continuous movement of zinc balls against the flow of the solution assures the contact of all the solution with zinc during its entire passage through the machine, and the

¹ "A New Method of Zinc Precipitation," *Eng. & Min. Jour.*, Jan. 27, 1917, p. 195.

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accumulation of all irregular and small pieces of zinc in the head of the machine with the precipitate. The separation of precipitate from solution is effected by discharging the machine into a

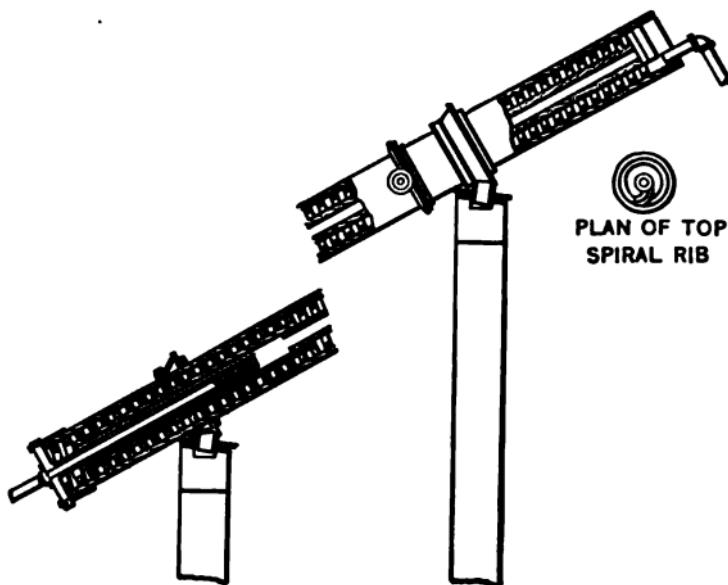


FIG. 23. Conklin Zinc-ball Precipitator.

steep conical tank, Fig. 24. The inner cones of this tank are of zinc, thus preventing any tendency of the precipitate to re-dissolve. The solution is discharged through bag filters attached to the bottom of the tank.

The following advantages are claimed for this machine:

- (1) No excess of zinc is used, but just enough.
- (2) Clean precipitate is produced, which can be melted with a minimum amount of flux, yielding the highest-grade bullion and saving the cost and loss of acid during treatment of the

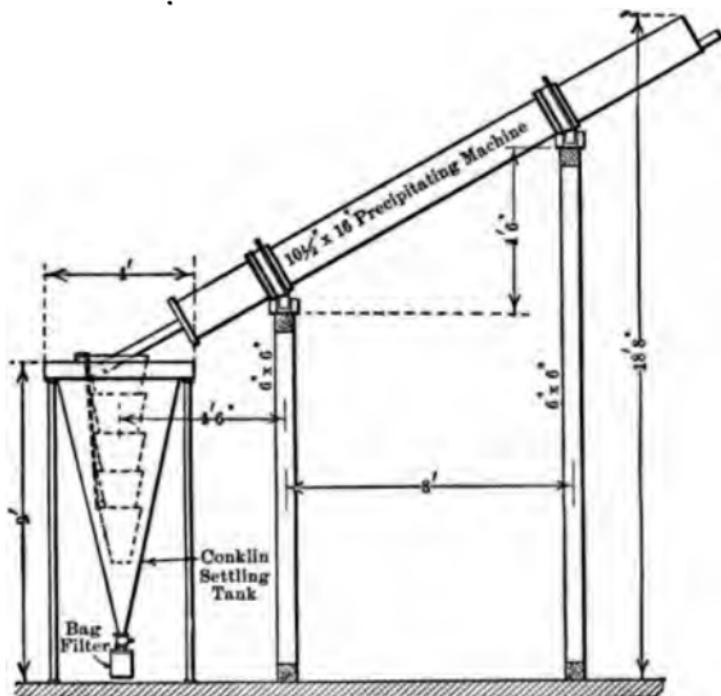


FIG. 24. Cone Settler for Conklin Precipitator.

precipitate. The inventor considers the machine practical.

Precipitation with Zinc Dust. — Zinc in the form of dust is considered to be superior to all other means of precipitation with zinc. This method is rapid, convenient, and gives a complete precipitation of precious metals. It was first proposed by H. L. Sulman, but was not successful at first, due to imperfect knowledge of the requirements necessary for its successful use. The important requirements are:

- (1) The zinc and precipitate should be removed from the solution immediately on completion of precipitation.
- (2) As far as possible, there should be no ~~.....~~

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tact between the zinc dust, in solution or out of it, with air, in order to avoid oxidation.

Merrill Precipitation Method. — The zinc powder, mixed with solution containing precious metals, is delivered into the suction of a solution pump, and passes through a filter press before being returned to the solution storage tanks. The pipe line from the pump to the storage tanks is usually quite long enough to permit the required time for thorough precipitation. Another system, devised by B. A. Bosqui, and now owned by the Merrill Metallurgical Co., is to mix the zinc dust with a measured quantity of solution in a tank, agitate by mechanical means for the required length of time to secure perfect precipitation, and then pump the mixture immediately through a filter press. In this way, little oxidation is undergone, and the time for precipitation can be regulated to suit all conditions. This method is used at Homestake.

The amount of zinc necessary for precipitation may be readily calculated from the known content of the solution in the tank. A quick method for estimating silver in solution has been devised by G. H. Clevenger¹ (see under Assay of Cyanide Solutions). The required quantity of zinc may be placed in a feeder, of which there are several types, and added to the solution continuously as the tank is emptied.

The advantages of the Merrill zinc-dust process are that the expense of the zinc-room, cutting of

¹ *Min. & Sci. Press*, May 13, 1913. Also *Puhasapa Quarterly*, Dec., 1915.

zinc, packing extractor boxes, and continuous watching are almost entirely eliminated. The presses are simply locked up and left until clean-up time. The clean-up always represents the entire production of bullion, and there is nothing left over in combination with zinc, as when shavings are used. Lastly, there is little danger of loss through theft or accident. Safety from accident is illustrated by the serious fire at the mill of the Goldfield Consolidated Co., at which time no precipitate was lost, although the presses were heated so hot as to burn the zinc in them almost completely.

Nature of Zinc Dust.—Zinc dust is the "blue powder" of commerce, which is unavoidably formed during the smelting of spelter. Its preparation consists merely in collecting the blue powder from the prolongs, as free as possible from admixtures of foreign elements, and sifting it. As its value depends upon its percentage of metallic zinc, it is essential, in preparing it for the market, to regulate the distillation and condensation of the zinc so that there will be the least possible formation of oxide. The market standard is a tenor of at least 90 per cent metallic zinc, but it is seldom so pure as this. Zinc dust is usually susceptible to oxidation, so much so as occasionally to be explosive, and on that account great care is necessary in its transportation and storage. Owing to its inflammable nature, and its liability to spontaneous combustion, it is rated by fire underwriters as a specially hazardous risk. A ruling of the Interstate Commerce Commission

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lists zinc dust as an inflammable product, and requires a special label and special packing in order to avoid breaking of the package.

Analysis of Zinc Dust. — Zinc dust should be dry and fine; nearly all, say 95 per cent, should pass a 200-mesh sieve (aperture 0.003 in.), while very little, say not over 1 or 2 per cent, should remain on 100-mesh (0.006 in.), and practically none on 50 or 60-mesh. It should not show any signs of caking, nor contain lumps which do not break up at once when shaken on a sieve. Generally an ash gray color is a more favorable indication than a decidedly blue or whitish cast, but color cannot be taken as a positive criterion. The presence of 2 to 3 per cent of lead is an advantage, giving more complete precipitation. Small percentages of zinc oxide are without detrimental effect. The contact of zinc with moisture should be avoided.

Determination of Zinc Oxide.¹ — Sherwood gives the following method based on the solubility of zinc oxide in ammonium and ammonia. The required solutions consist of 250 c.c. water; 70 gr. ammonium chloride; and 150 c.c. strong ammonia water (sp. gr. 0.90). Weigh out 1 gr. of zinc dust, put into a stout test tube holding from 35 to 50 c.c. Use a disk of soft-rubber packing, $\frac{1}{8}$ in. thick, as a cover while shaking. Add 25 c.c. of the prepared solution; cover tube and shake well for just 5 min. Agitation must be violent enough to insure all lumps being broken up.

¹ "Tests of Zinc Dust for Cyaniding," by W. J. Sherwood. *Eng. & Min. Jour.*, Vol. 93, p. 944.

Throw mixture promptly on a 9 or 11-cm. filter, rinse tube and wash filter with hot water, receiving in a 200- or 250-c.c. beaker. Dilute filtrate with hot water to about 100 c.c. Add a drop of phenolphthalein indicator, then HCl until about neutral, then 5 c.c. strong hydrochloric acid. Heat to 80° C. and titrate with ferrocyanide, using uranium acetate or nitrate as indicator. Zinc found in solution \times 1.245 = zinc oxide in 1 gr. This method is rapid and gives fairly approximate results if carried out as described. Results will uniformly be slightly high due to the fact that metallic zinc is also slightly dissolved by the solvent.

*Determination of Zinc.*¹—One gram of zinc dust is weighed into a dry stoppered 200-c.c. flask, mixed with 100 c.c. potassium-dichromate solution (30 gr. per liter) and 10 c.c. of 1:3 sulphuric acid, and agitated for 5 min. Another 10 c.c. of acid is then added and the shaking continued for 10 or 15 min., when everything, except a small earthy residue, should be dissolved. The liquid is diluted to 500 c.c., and in 50 c.c. thereof the excess of dichromate is determined by introducing 10 c.c. of 10-per cent potassium iodide and 5 c.c. of sulphuric acid, titrating the liberated iodine with N/10 thiosulphate.

A blank test should be made by mixing 100 c.c. of standard dichromate with 20 c.c. of H₂SO₄ (33 per cent by volume), diluting to 500 c.c., mixing thoroughly, and taking out 50 c.c. Mix

¹ Sutton, "Volumetric Analysis," 10th ed., p. 383, quoted for "Analyst," 25, 279.

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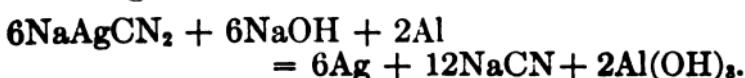
with 5 c.c. of 33-per cent H_2SO_4 and 10 c.c. of 10-per cent KI, and titrate with N/10 thiosulphate. The difference between the amount of thiosulphate required in the blank test and that consumed in the test on the zinc sample gives the equivalent of dichromate reduced by the zinc.

For the determination of minor constituents in zinc dust the reader is referred to "Notes on the Analysis of Zinc Dust," by J. E. Clennell, *Eng. & Min. Jour.*, April 19, 1913.

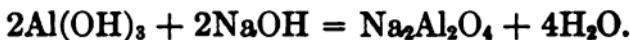
PRECIPITATION WITH ALUMINUM.

The use of aluminum as a precipitant is of comparatively recent date. It has been in use for the past 5 or 6 years in a few mills, but until recently its advantages have been only imperfectly recognized. An attempt to use aluminum in the form of shavings developed the fact that when a light and coherent coating of metal was formed on the shavings, precipitation stopped, and a large amount of aluminum went to waste. In 1910 S. F. Kirkpatrick used aluminum in the form of dust, and thus avoided the difficulties experienced with plates or shavings. When used as dust, the metallic particles are entirely dissolved before any plating action can occur.

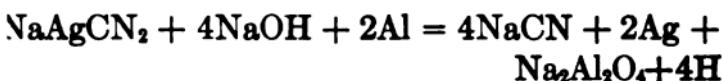
Reactions. — The fact that aluminum does not replace the precious metals in a cyanogen compound renders necessary the presence of caustic soda or potash, the reaction probably occurring thus:



he aluminum hydroxide at once dissolves in the excess of caustic to form sodium aluminate:



if this equation is correct it should be possible to make one part of aluminum precipitate 12 times its weight of silver, but in practice it is found to precipitate only about 3 times its weight. Hence, with low-grade solution, when caustic and aluminum are present in excess, the following may more nearly represent what actually occurs:¹



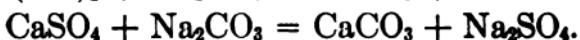
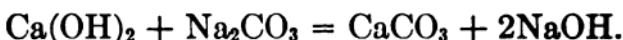
Requirements for Aluminum Precipitation.—Although the cost of aluminum and of caustic soda are now extremely high (before the war aluminum dust cost 24¢ per pound and caustic soda about 2.11¢, present prices being about 90¢ for aluminum and 5.77¢ for caustic soda) the use of aluminum dust may be almost necessary in some cases. For example, at the mill of the Butters Divisadero Co., the material is of such nature that the gold solutions once passing over it, if used again, rapidly lose their dissolving efficiency, and after the second precipitation are less efficient than after the first. Since it is practically certain that aluminum forms no compounds with cyanogen, not only is the whole

¹ "Aluminum Precipitation at the Mill of the Butters Divisadero Co.," by E. M. Hamilton and P. H. Crawford, *Min. & Sci. Press*, Sept. 11, 1915. Also *Mineral Industry*, 1916.

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of the cyanide recovered which was combined with the precious metals, but also the additional loss of cyanide by direct combination with zinc is avoided. In the handling of strong solutions the latter may be a considerable factor.

Maintenance of Alkalinity. — With aluminum precipitation, an ore of high acidity and of a sliming nature may cause trouble, since lime must be added both for protective alkalinity and for settling slimes. It is essential, however, that lime shall be absent from the solution at the time of precipitation, because it reacts with sodium aluminate and is precipitated in the press as calcium aluminate, yielding at the clean-up a product which is almost impossible to melt. The cost of substituting caustic soda for lime is prohibitive, and the amount of lime cannot be regulated just to neutralize the acid present. This condition has been overcome by a method devised by Hamilton.¹ The solution before precipitation is treated with soda ash, throwing out the lime as carbonate, and generating caustic soda:



This method allows the use of lime for neutralizing and also for settling slime, it yields a lime-free solution for precipitation, and utilizes a cheap source of alkali.

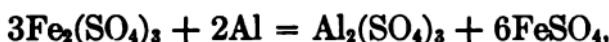
Estimation of Metallic Aluminum in Aluminum Dust. — The value of aluminum dust for technical purposes depends chiefly upon the

¹ Patented, April 20, 1915.

ESTIMATION OF ALUMINUM IN DUST 157

amount of metallic aluminum contained. The following method devised by J. E. Clennell¹ is quite rapid and has given satisfactory results. Weigh accurately 100 mg. of aluminum dust and place in a conical flask provided with stopper and Bunsen valve to reduce risk of oxidation, together with 5 gr. of pure ferric sulphate, roughly weighed. Add 50 c.c. of dilute sulphuric acid, containing 25 per cent H_2SO_4 by volume, and heat the mixture gradually until the ferric sulphate has completely dissolved and the liquid is boiling gently. Usually by the time this condition is reached, the whole of the metallic aluminum will also have dissolved; a few fragments occasionally require longer treatment. If the aluminum dissolves completely with moderate heat, it is not essential to boil. Cool the flask to room temperature by placing under a running tap, and add 50 c.c. of cold distilled water. The liquid is then titrated with standard permanganate solution containing 3.35 gr. $KMnO_4$ per liter; 1 c.c. = 1 per cent Al, approximately.

Since the method is based upon the reduction of ferric sulphate by metallic aluminum,



and the titration back to the ferric condition is made with permanganate, no hydrogen must be liberated by the reaction of aluminum with sulphuric acid; this may, however, take place, in which case the amount of ferrous sulphate

¹ Eng. & Min. Jour., May 6, 1916.

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formed would not be proportional to the amount of metallic aluminum present.

Another source of error, acting in the opposite direction, is the presence of iron, zinc, or other metal capable of reaction with ferric sulphate. For accurate work, therefore, these metals must be determined and their effect allowed for. When, however, comparative results only are required, as in checking the quality of factory products from day to day, the rapid test described is sufficient.

In making the ordinary tests for controlling plant operation, a sample of the purest obtainable aluminum dust is accepted as a standard. Tests are then made concurrently with the standard dust and with the sample to be examined, the result of the latter being expressed as a percentage of the standard.

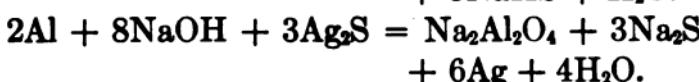
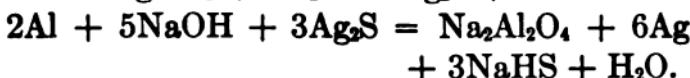
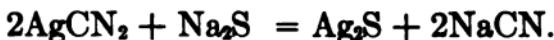
SODIUM SULPHIDE PRECIPITATION.

Nipissing Practice. — The following method, as used at Nipissing,¹ was developed on account of the increase in price of aluminum, which had been used as a precipitant. In the course of investigation, two objections in the use of sodium sulphide were made apparent: First, that complete precipitation could not be obtained without using an accessory reagent; and second, the tendency to produce silver sulphide instead of

¹ "Sodium Sulphide Precipitation at Nipissing," by R. B. Watson, *Eng. & Min. Jour.*, Mar. 3, 1917. Also *Trans. Canadian Institute of Mining Engineers*, Nov., 1916.

metallic silver. As to the first point, it was found that the solution could be precipitated to 0.2 oz. silver without an excess of sodium sulphide in the barren solution. The second objection was met by converting the silver sulphide to bullion by the same method employed in the desulphurization of raw ore before it is sent to cyanide treatment. That method is based on the fact that the various sulphides and combinations of silver are decomposed by contact with metallic aluminum in a caustic soda solution. This is an important feature of the process, since it gives fine bullion at small cost.

Reactions. — In the former days of the sodium hyposulphide process, before silver ore was treated by the cyanide process, the treatment and marketing of the sulphides produced were the principal obstacles. The reactions in the precipitation of silver from cyanide solution by sodium sulphide are:



The commercial sodium-sulphide employed is supposed to be $\text{Na}_2\text{S} + 9\text{H}_2\text{O}$, equivalent to 32.5 per cent Na_2S .

From the equation it appears that 0.076 lb. of commercial sodium sulphide would be required to precipitate 1 oz. of silver; in practice, the amount required is 0.12 lb. In the desul-

phurizing, 1 oz. of silver requires 0.021 lb. of NaOH, equivalent to 0.028 lb. of commercial caustic soda carrying 76 per cent NaOH. If an excess of caustic is present in the operation, the reaction in the third equation might take place, in which case 0.034 lb. of NaOH is consumed for each ounce of silver desulphurized. To prevent this, the amount of silver present in the charge to be desulphurized is estimated as closely as possible, and 0.03 lb. of commercial caustic is added for each ounce of silver contained. The theoretical amount of aluminum necessary to desulphurize 1 oz. of silver is 0.0057 lb.; the amount actually used in practice being 0.006 lb.

Practical Details of the Process. — The procedure consists in dissolving the sodium sulphide in an iron barrel into which is flowing constantly a small stream of barren solution from the filter press. The barrel is divided by a vertical partition reaching nearly to the bottom. There is an excess of sodium sulphide in the barrel, and the overflow pipe leaving the side opposite from the inflow carries the precipitant to a tank, where it is agitated with pregnant solution, and both are then pumped to a Merrill filter press. The amount of sodium sulphide added is regulated by a boy, who constantly tests the inflowing barren solution with pregnant solution and with sodium sulphide solution to determine whether there is an excess of sodium sulphide or of unprecipitated silver in the discharge from the press. The amount of barren solution coming into the barrel, and consequently the overflow of sodium

sulphide, is regulated by a valve on the intake pipe in front of the operator. This operation could be carried out more precisely, perhaps, by precipitating a tankful at a time. The solution from the filter press is aerated thoroughly in the barren-solution tank by air lift and a large circulating pump, the object being to oxidize any slight excess of sodium sulphide in the precipitated solution. The shiftman always tests the solution for soluble sulphide before running it to the treatment tank, and no bad effect has been noticed from this cause. Aeration is used in preference to lead salt, as lead has a bad effect on extraction.

A tankful of sulphide precipitate contains 25,000 oz. of silver. This is drawn into the tank and the proper amount of caustic soda is added, 0.03 lb. for each ounce of silver. The dilution is four parts of solution to one of sulphide, and the strength of the solution in NaOH is about 8 per cent. The pulp in this tank is kept mixed by the usual mechanical stirring device. Over the tank is mounted a small tube-mill, which makes 10 r.p.m. The charge of aluminum ingots weighs 300 lb. and fills the tube-mill half full. Below the tank is a centrifugal pump which circulates the pulp from the tank through the tube-mill for 10 or 12 hours. When desulphurization is about complete, the precipitate rubbed on a piece of paper by a spatula gives a silvery color and a metallic luster. It is then pumped to another press, where it is washed for two hours; the resulting product, then mixed with a small amount of water, is charged, without drying, to the rever-

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beratory furnace, and melted down to bullion, which comes out about 996 fine. The slag is saved and put through a small blast furnace.

The grade of the precipitate depends largely on how well the pregnant solution has been clarified before precipitation, but under ordinary conditions it assays about 23,500 oz. before and 26,500 oz. after desulphurization. The final product carries from $\frac{1}{2}$ to 1 per cent sulphur. Gold is not precipitated in an alkaline solution, and copper does not come down if there is sufficient free cyanide in the solution.

Considering the present war price of aluminum dust, the continued use of this reagent for precipitation is out of the question. It would appear, however, that even when prices of all chemicals return to normal, sodium sulphide precipitation will be cheaper than the method formerly employed.

ANALYSIS OF PRECIPITATE.

	Before desulphurizing.	After desulphurizing.
	Per cent.	Per cent.
Ag.....	81.21	91.72
S.....	12.09	0.58
Soluble in water.....	2.02	0.52
Insoluble in acid.....	2.63	3.03
FeO.....	0.46	0.57
CaO.....	0.44	0.49
Al ₂ O ₃	1.32	1.55
MgO.....	0.14	0.16
Hg.....	0.10	0.11
	100.41	98.73

The precipitate contained no Au, Pb, Cu, Bi, Co, Ni, Sb, As, or Zn.

PRACTICAL DETAILS OF THE PROCESS 163

In the following comparison pre-war prices are given; the quantity of aluminum dust used per ounce silver is the average for 1915.

COST OF SULPHIDE COMPARED WITH ALUMINUM PRECIPITATION.

Aluminum Dust Method.

	Cents per oz silver.
0.0262 lb. aluminum dust @ 33.82¢	0.8861
0.05 lb. caustic soda @ 2.11¢	0.1055
Labor, power, and workshops	0.1710
	<hr/>
	1.1626

Sodium Sulphide Method.

0.12 lb. commercial sodium sulphide @ 1.89¢	0.2268
0.006 lb. aluminum ingots @ 19.25¢	0.1155
0.03 lb. caustic soda @ 2.11¢	0.0633
Labor, power, and workshops	0.3020
	<hr/>
	0.7076
Difference in favor of sodium sulphide method	0.4550¢
Saving per year on 2,000,000 oz. @ 0.45¢	\$9,100

It was shown in describing Hamilton's process that much of the advantage in the use of aluminum over zinc dust is in the regeneration of cyanide; it will be noted that regeneration also occurs when sodium sulphide is used. The process, as described, is limited to silver ores, as gold sulphide is not precipitated. It is possible, however, that the method could be used to precipitate the silver, and be followed by the precipitation of gold from the solution by aluminum.

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ELECTROLYTIC PRECIPITATION.

The present condition of the market for zinc and aluminum, especially the latter, the present cost of which has caused many plants to abandon its use as a precipitant, has caused considerable emphasis to be laid upon precipitation by electrolytic means; this is also a most feasible substitute at any time.

We have seen that a great variety of compounds results from the decomposition of cyanide, as well as from its combination with elements in the ore treated, or introduced by chemical methods of precipitation. Among these, the principal constituents of such solutions, if zinc precipitation were not used, would be simple alkaline cyanides, alkaline hydrate, gold and silver, and copper, if it occurs in a soluble form in the ore undergoing treatment. The proportion of these various constituents, particularly of gold and silver, is extremely small as compared with the amounts of metal present in solutions made by extraction processes in other industries; for example, in a solution containing 0.50 oz. of gold per ton, the gold would form only 1/58,332 part of the solution; in a solution containing 0.05 oz. gold per ton the gold would constitute only 1/583,320 part. In the case of waste solutions, which should be precipitated before they are discarded, it is not unusual to deal with solutions containing less than 0.01 oz. of gold per ton (20¢), or only 1/2,216,600 of the total weight of the solution. Tailings solutions containing only 1 or 2¢ per

ton are common at many mills using zinc-dust precipitation.

The electrolytic precipitation of gold and silver from cyanide solutions, therefore, involves the total precipitation of an extremely small weight of metal from solutions containing a comparatively small proportion of soluble salts, which are, therefore, poor electrical conductors. The fact that the alkaline cyanides are readily decomposed by electrolysis, and that the economy

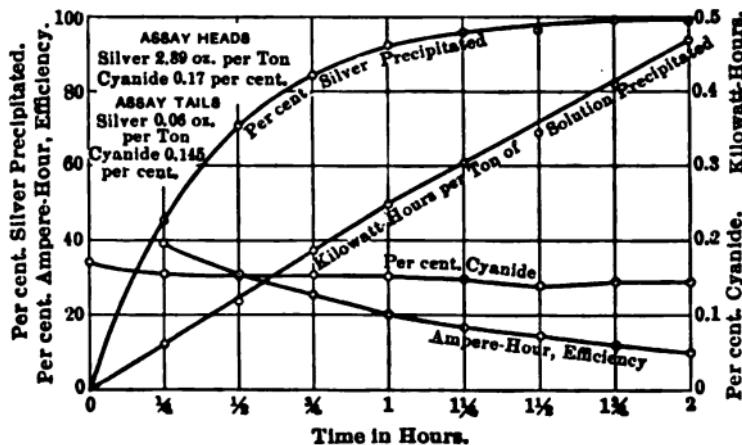


FIG. 25. Precipitation from Fresh Cyanide Solution.

of the cyanide process depends upon the continued re-use of the solvent, makes it imperative to precipitate with minimum decomposition of cyanide, and preferably with a regeneration of cyanide.

It is well known that as the proportion of metal in solution decreases, the difficulties of precipitation increase. Thus it may require as great an electrode surface, as long a time, and as great a consumption of power to precipitate the 1-

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10 per cent of gold and silver as the first 90 per cent. This is illustrated by the curves in Figs. 25 and 26,¹ which embody the results of small-scale experiments, during which the solution was agitated throughout electrolysis by a small wooden propeller driven by an electric motor. Fig. 26 shows the results upon a fresh cyanide solution

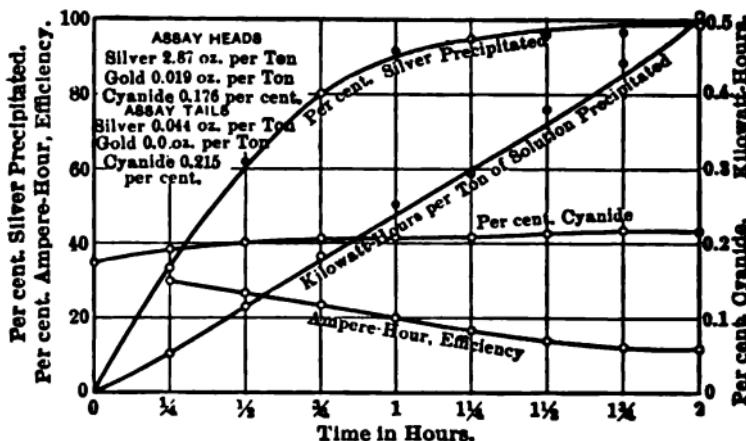


FIG. 26. Precipitation from Tonopah Extension Mill Solution.

containing silver; the loss of cyanide during precipitation is apparent. Fig. 27 shows the results upon Tonopah Extension mill solution of very nearly the same silver content. In this case a gradual increase in the cyanide strength up to the end of the test is to be noted. This is to be expected of a mill solution containing zinc and sulphocyanate.

Rapidity of Electrolytic Precipitation.—Metals in solution can be precipitated by elec-

¹ "Electrolytic Precipitation from Cyanide Solutions," by G. H. Clevenger, *Eng. & Min. Jour.*, Sept. 30, 1916.

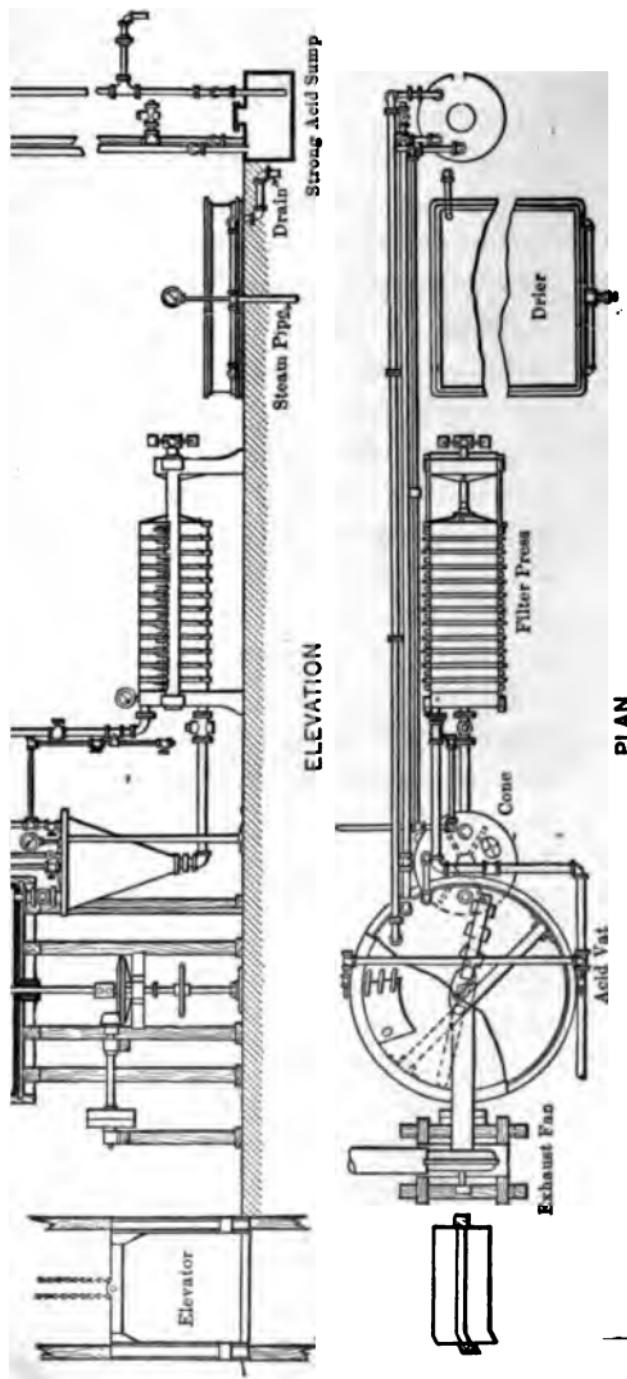


FIG. 27. Homestake Acid-treatment Plant.

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trolysis only as fast as they are brought in contact with the electrodes. When it is remembered that it is only from the thin film of solution in contact with the cathode that precipitation is taking place, the necessity for agitation or rapid movement of the solution, in order that the thin film of impoverished solution may be quickly replaced by solution containing the metal, is apparent. This becomes of greater importance as the metal content of the solution decreases. In most early installations this necessity was overlooked; the invariable practice was to allow the solution to pass gently by gravity, flowing through the compartments in which the electrodes were suspended. In general, the agitation of the solution is very important.

Anodes.— The anodes have always been a source of weakness in electrolytic precipitation. The precipitation of gold and silver from cyanide solutions, employing a soluble anode of the metal to be deposited, had been in practice in electroplating long before the complete precipitation of gold and silver from dilute solutions resulting from ore treatment was attempted by electrolysis, using an insoluble anode.

Plants operating the Siemens-Halske process invariably use an iron anode. This anode has the disadvantage of rather rapid disintegration unless the current density be low. This results in the formation of either prussian blue or ferric hydroxide, the former being particularly objectionable.

Many plants use lead or preoxidized lead

anodes. With lead anodes a reasonably high current density can be used, and they avoid the formation of troublesome iron compounds. A lead anode has a long life, and when finally discarded, has a salvage value for lead and other metals that it may contain.

Preoxidized lead, on the whole, has been found most satisfactory. Under ordinary conditions, and when properly prepared, it has a life of one year, and then has a salvage value. When pre-oxidizing lead plates, they must first be cleaned and thoroughly brightened with a scratch brush. At Minas Prietas a 1-per cent potassium permanganate solution was used, with a current density of 1 amp. per sq. ft.; the time was 1 hr. During the operation the strength of the solution was frequently tested with a standard solution of ferrous sulphate, fresh permanganate being added to keep the solution as near 1 per cent as possible.

Hard carbon was early tried upon a large scale by von Gernet, but was abandoned because in the course of time it formed compounds with cyanide and disintegrated. Acheson graphite has also been tried upon a working scale; though this material gives splendid results with certain electrolytes, it does not seem to be suitable for cyanide solutions, for in the course of a short time it becomes soft and mushy, and finally useless.¹ This is probably due to disintegration of the binding material used in the original carbon plates from which the graphite plates were made. Furthermore, graphite has no salvage value.

¹ Clevenger, last reference.

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High-silicon cast iron has been suggested, because it has been so satisfactory in the chemical industries for resisting the corrosion of acids.

Fused iron oxide or magnetite electrodes have been used experimentally as an insoluble anode material in various electrolytic industries in this country. The large electrolytic precipitation plant for copper at the Chuquicamata mine in Chili was originally equipped with these anodes. More recently a large number of them have been replaced on account of the difficulty of obtaining magnetite electrodes. The cost of this material is rather high; it is also very brittle, and in practical operation its life would not be indefinite, as has sometimes been claimed.

Cathodes. — Sheet lead is the material most used for cathodes, but other materials, such as tin plate and aluminum, have been used. Aluminum has its advantages when a noncoherent deposit is desired, but its cost and its tendency to disintegrate in an alkaline solution when the current is off, are drawbacks. Sheet lead with a noncoherent deposit will not stand the brushing and scraping on account of its fragile nature.

Electrode Area. — One serious objection to electrolytic precipitation has been the "large electrode surface" required, which necessitates large and unwieldy units, especially as compared with the plant required for zinc- or aluminum-dust precipitation. Several ways are suggested for diminishing the requisite electrode surface. With active agitation, the current density could be considerably increased and a reasonable efficiency

PRECIPITATION WITH CHARCOAL 171

maintained. Since the amount of salts carried in ordinary cyanide solutions is comparatively small, the conductivity of the solution would be increased by adding a certain proportion of some cheap inert salt; this, of course, would not be without its complications, for it would mean welcome chemical reactions.

The only other way that the current density can be increased is by increasing the voltage or by spacing the electrodes closer together. Increase in voltage, however, increases the liability to short circuit, and the power consumption. In practice, the electrodes are placed about 1.5 in. apart; owing to the tendency of electrodes to stick together, any closer spacing will result in short circuiting. Voltage between anode and cathode varies from 2.5 to 8; current density at cathode varies from 0.03 to 1.20 am. per sq. ft.

In electrolytic precipitation, by-products cause much trouble and are always a serious problem. In the Siemes-Halske process, the Prussian blue, resulting from the attack on the iron anodes, may require special treatment for the recovery of silver and gold. Calcium salts also separate and deposit upon the electrodes.

PRECIPITATION WITH CHARCOAL.

Two recent articles by W. R. Feldtmann¹ and F. J. Green² have gone far to clear up the

The Precipitative Action of Carbon in Contact with Aqueous Cyanide," *Min. & Sci. Press*, May 22, 1915, I. of M. & M., No. 127, p. 15.

² *Trans. I. of M. & M.*, Vol. XXIII, p. 35.

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mystery surrounding the action of charcoal and similar matter on gold in cyanide solution.

The precipitation of gold by charcoal has been recognized for many years and has generally been attributed to some special peculiarity of amorphous carbon which it would be hard to classify with any other known phenomenon. But the work described in the above-mentioned papers seems to offer, from the colloidal standpoint, an explanation as to how gold is adsorbed into charcoal and why it does not behave like ordinary adsorptions.¹

Mr. Green has clearly shown that charcoal containing adsorbed carbon monoxide is capable of precipitating gold; that charcoal which has been exhausted in a high vacuum at elevated temperatures possesses small precipitating power but that this power is restored by allowing the CO to be re-adsorbed, or by heating the charcoal in air. This is readily explained, if CO is the active agent in precipitation of gold, because a simple heating in air generates more CO in the pores of the charcoal; it also explains why freshly heated charcoal has a greater precipitating power than charcoal that has stood for some time,

¹ "Since the property of adsorption of ions by colloids has been firmly established, there have been more or less serious attempts to find an explanation for the precipitating action of carbon in cyanide solution, but those doing the work have either been unsuccessful or forced to assume that the adsorption of gold-bearing ions in the pores of charcoal was of a slightly different kind than the adsorption of ions by other colloids, or of other ions in charcoal itself."

the former doubtless being saturated with carbon monoxide, while the latter has lost part of it by diffusion. Carbon belongs to a class of amorphous substances such as meerschaum, colloidal gels, starch, cotton, wool, glass, coal, etc., which have this property of adsorbing on their surfaces great amounts of gases, vapors, liquids, and even ions from solution.

The following is an analogy between the precipitating action of West African carbonaceous schists and ordinary wood charcoal, both being in contact with auriferous cyanide solutions.

1. Gold is precipitated by both (in the case of schist, in a measure roughly proportional to the amount of carbon present).

2. The gold thus precipitated is not soluble to any appreciable extent in fresh cyanide solution.

3. It is soluble to a larger extent in alkaline sulphides.

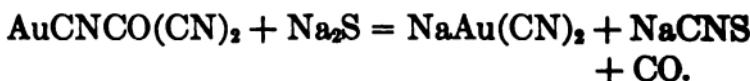
4. After dissolving the gold in alkaline sulphides the precipitating capacity of both schist and charcoal is renewed practically in its entirety.

According to Feldtmann, precipitation by charcoal from cyanide solutions is quite a different action from the precipitation of gold from gold chloride solution. (a) Charcoal rapidly and "visibly" precipitates "metallic" gold from gold chloride solution in a form which can readily be dissolved in weak cyanide solutions, but is quite insoluble in alkaline sulphide solutions. (b) From alkaline aurous cyanide solutions, of equal value, gold is precipitated comparatively slowly in

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a form which is not visibly metallic (indeed there is no visible coating of any sort on the charcoal). Gold thus precipitated is not soluble in fresh cyanide solutions, but is soluble to a considerable extent in alkaline sulphide solutions.

Feldtmann adopts as a working formula for gold as precipitated by charcoal, "carbonyl aurocyanide," $\text{AuCnCO}(\text{CN})_2$, and represents its solution in alkaline sulphide, as:



the CO being re-adsorbed¹ and the precipitating power of the charcoal renewed (as, in great part actually happens). The precipitation of gold from sulphide solutions appears to be best effected by contact with metallic copper.

¹ " Adsorption " is the name applied to the phenomenon exhibited by colloid substances, which are able to take dissolved substances out of solution, and hold them, often with great tenacity. Some substances exhibit selective adsorption; thus seaweed adsorbs iodine from sea water, although the quantity present is so small that it cannot be detected by any of the usual tests. Adsorption is a surface tension phenomenon; increased concentration of the adsorbed surface on the boundary of the colloid particles leads to decreased surface tension. In a few cases increased concentration leads to increased surface tension and negative adsorption is then exhibited.

CHAPTER XII.

TREATMENT OF PRECIPITATES.

Mint Requirements. — The different methods of precipitating gold and silver from cyanide solution were described in Chap. VIII. In the operation of precipitating with zinc, in whatever form, the resulting precipitate will contain more or less zinc, which must be removed with other impurities to bring up the grade of bullion as high as possible. The question arises, to what extent is it profitable to refine bullion for the mint? The regulations of the mint and assay offices of the United States, in effect on April 15, 1916, provide the following requirements:

(A) Bullion containing over 800/1000 base will be declined.

(B) When bullion is so impure that the loss in weight in melting exceeds 25 per cent, a charge in addition to the regular charge is made.

(C) Bullion which does not give concordant assays on the first dip may be charged extra for remelting and treatment; and if the second dip fails to give concordant assays the bullion will be refused.

(D) The regular charges are three in number, as follows:

1. *Melting Charge.* — Equal to \$1 for each 1000 oz. or fraction thereof, as determined from the weight of bullion after melting.

2. *Parting and Refining Charge.* — This is the main item of the charges and depends upon the fineness of the gold and the fineness of the doré. From the following table the charges for parting and refining per ounce of weight after melting, can be determined for each fineness of gold and doré.

Gold, fine.	1-250.	250½-500.	500½-949½.	950-991½	992-1000.
Doré, fine.	Charge per ounce weight after melting.				
290-299	\$0.08	\$0.09
300-399	0.07	0.08
400-499	0.06	0.07
500-599	0.05	0.06	\$0.04
600-699	0.04	0.05	0.04
700-799	0.03	0.04	0.04
800-899	0.02	0.03	0.04
900-969	0.01	0.02	0.04	\$0.04
970-1000	0.01	0.02	0.04	0.02	No chrg.

3. *Alloy Charge.* — Equal to 2.5¢ for each ounce of copper required for alloy, as determined by taking one-tenth of the fine weight of gold in the bullion.

Removal of Undissolved Zinc. — At a gold cyanide mill, on clean-up, all material passing a 10- or 20-mesh screen is treated, that remaining on the screen being returned to the zinc-boxes. In a silver mill, owing to the smaller value of the silver bullion, only the slime passing through a 30- to 60-mesh screen is usually taken, all the short zinc being returned to the box. It may be advisable to screen the precipitate into two classes: ~~that~~ held on a 30- to 60-mesh screen, high in

zinc and low in bullion, and that passing the screen, which would be high in bullion and low in zinc, so that separate treatment may be given to each.

Treatment of Zinc Precipitate. — The precipitate may be refined into bullion by four different methods:

1. Melting.
2. Roasting and melting.
3. Acid treatment and melting, with or without roasting.
4. Smelting with litharge and cupellation.

Melting. — When the precipitate is simply to be melted, it is first dried completely, or as nearly completely as possible, by a small filter press which draws air through the precipitate and removes most of the moisture. The dried precipitate is mixed with the proper flux and melted. This method is suitable for material passing a 40- to 60-mesh screen, which is high in gold and silver.

Roasting. — By roasting, the metallic zinc is converted into oxide, in which form it more easily enters the slag either a silicate of zinc or mechanically included oxide. Other base metals are oxidized, or volatilized, also putting them into a more suitable form for removal by slagging. Roasting is recommended for coarser precipitate containing much zinc. The extent of roasting is best gauged from experience. Roasting is often carried to the point where the zinc takes a dull fire; in this case, it should be conducted slowly, so that the zinc shall oxidize instead ~~as~~

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volatilizing, and the fumes and ebullition shall not carry off gold and silver. To assist in the oxidation of zinc and other impurities, nitre, KNO_3 , to the amount of from 3 to 10 lb. is added to the precipitate, in the form of powder or as a solution; zinc is thus more rapidly converted to oxide, in which form it does not volatilize. The precipitate, while roasting, should be stirred as little as possible, to prevent loss from dusting.

Acid Treatment. — The precipitate is settled and dewatered by decantation in the sludge tank, into which it runs from the zinc-boxes, and acid is added to dissolve the zinc. The acids that have been used for this purpose are: Sulphuric, H_2SO_4 ; sulphurous, H_2SO_3 ; hydrochloric, HCl ; and bisulphate of sodium, NaHSO_4 . Nitric, HNO_3 , has been used, but it is not to be recommended, as its solvent action upon precious metals is high. Hydrochloric acid is not generally used, for the same reason, although it has the advantage of forming soluble chlorides with lime or lead, which can then be removed by washing. Acid treatment is common with gold precipitates, but is rarely attempted for treating silver precipitate, as all zinc not passing 40- to 60-mesh screen is usually returned to the zinc-boxes; it would also cause a loss by dissolving much silver.

Sulphuric Acid Treatment. — After the water has been decanted from the settled slime in the sludge or clean-up tank, which should be of wood or lead lined, if acid treatment is to be used, sulphuric acid is added carefully to avoid boiling over of the charge, in an amount to make from



to 20-per cent solution of acid. Theoretically, one part of zinc requires 1.5 parts of sulphuric acid to convert it to zinc sulphate, while a 16-per cent solution of acid appears to act to the best advantage on zinc; in actual practice, from 75 to 1.75 lb. of cheap commercial sulphuric acid is used for each pound of dry precipitate. A hood should be placed over the treatment vat to remove hydrogen and hydrocyanic acid, which are formed in the treatment. When danger of boiling over is past, the sludge may be stirred by hand or by mechanical means; a sudden tendency to boil over is allayed by the addition of a small amount of cold water. Sulphuric acid is added from time to time until action ceases and an excess of acid is present, the zinc having all been dissolved. This operation of "cutting down" the precipitate may be accomplished in a couple of hours, but generally an entire shift is allotted to it.

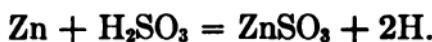
The operator must be cautious not to be overcome by fumes. Ordinarily the breathing of ammonia fumes will afford relief. If the ore contains arsenic, fumes of arseniureted hydrogen may be given off which are highly poisonous; several deaths have occurred from this cause. Preliminary treatment with nitric acid, or a general treatment with one part nitric and two parts sulphuric acid, to change the arsenic into a non-volatile arsenic acid, has been recommended; so treatment with bisulphate of sodium, although it would appear better to dispense with acid treatment entirely on such a precipitate.

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When the zinc is completely dissolved, the tub is filled with water, the sludge well stirred and allowed to settle, and the clear solution decanted off. After repeating several times, the sludge is conducted into a small filter press, or vacuum filter, where it is washed thoroughly by drawing hot water through it, which removes all sulphates and other soluble matter. One part of water at 0° C. will dissolve 0.42 part of zinc sulphate; at 20°, 0.53 part; at 50°, 0.67 part; and at 75°, 0.80 part. Lead sulphate is practically insoluble, and 1 part of calcium sulphate is soluble in 500 parts of water. The solubility of calcium chloride is 1 part in 1.33 of water, and of lead chloride 1 part in 93 parts of water; this indicates the advantage of using hydrochloric acid when the precipitate contains large quantities of lime or lead. Lead and calcium sulphates, aside from contaminating the acid-treated slime, may coat the zinc so that it is not readily dissolved.

The acid washes should be collected in a tank and allowed to settle until the next clean-up, when they are siphoned to waste; or, they may be agitated with scrap zinc before settling, or run to waste through such zinc. The partly dried slime is removed from the filter press or vacuum tank and dried (electric driers are convenient); it is then roasted, or fluxed and melted in the damp condition. Acid treatment, followed by a thorough roasting, is the method usually practiced in this country on gold slime, while silver slime, with or without roasting, is melted without acid treatment.

sulphurous Acid Treatment. — The acid employed for this purpose is made by burning sulphur in a generator, or air-tight stove, supplied under a few pounds of pressure. This gives the necessary oxygen to form sulphur dioxide and forces it into a clean-up tank where it is absorbed by water, forming sulphurous acid. The precipitate may be already in the clean-up tank or may be added after formation of the acid. The zinc dissolves as:



In this treatment, the sulphites of lead and zinc are formed, which, like the sulphates, are soluble in water. The advantages of this method are: 1, Low cost; 2, greater ease and safety in transporting sulphur rather than sulphuric acid. This method has been successfully used in the United States.

Ammonium Bi-sulphate Treatment. — The chemical treatment is carried out in a stock tank, and is used to dilute the treatment solution until it contains 10 per cent H_2SO_4 . The dissolving zinc reacts according to the following equation:



It is to be noted that this method is accompanied by a danger from poisoning by arseniureted hydrogen. The advantages attending use of this acid would apply to this method.

Fluxing and Smelting. — The nature of the fluxes to be used will naturally vary at different plants, according to the

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method of precipitation; the amount of base metals in the ore; the practice of cleaning-up and refining; and whether the principal value of the ore is silver or gold. The main constituents of precipitate are the following:

- (a) Gold and silver.
- (b) Metals and bases in the form of oxides and sulphides.
- (c) Metals in native form, and other reducers.
- (d) Silica or lime.

The amount of base metal to be slagged off depends upon the manner in which the precipitate has been cleaned, especially the mesh of the screen it has been put through; also whether or not it has been acid treated before roasting. Roasting ordinarily changes base metals to oxides; sulphates formed while cutting down with acids, and not removed by washing, may remain as sulphates. Zinc may be present in metallic form, and also lead if a zinc-lead couple has been used. Silica will occur when dirty solutions are allowed to pass through the zinc-boxes. Lime and other alkaline substances may be deposited in the boxes when the protective alkalinity is too high.

Purpose of Fluxing and Smelting. — The purpose of these operations is to produce fine bullion, by eliminating the base metals in the form of slag and dissolved oxides. It is necessary to add the proper kind and amount of flux to unite with the base metals and form a slag of such fluidity that the gold and silver globules can settle through it to the bottom of the crucible. The slag also should form and remain liquid at

as low a temperature as possible; and it should be of such chemical nature as not to attack the crucible unduly. An acid slag will be less corrosive on the crucible, but it may be too viscous, and does not possess the power of dissolving oxides of base metals to the same extent as a basic slag. The fluxing reagents generally used are: Sodium carbonate, potassium carbonate, borax and borax glass, manganese dioxide.

Table IV gives the nature and chief properties of the common fluxes.

Determining the Flux to be Used. — The correct fluxing of precipitates must be worked out for each mill. One method is to prepare test charges of precipitate and flux, smelt them in assay crucibles in the assay furnace, and observe the resulting slag and button, probably assaying or panning the slag to find how low in values it is. This method cannot be entirely relied upon, and when preparing a flux for the precipitate at a new plant, experience and careful observation of the precipitate must be depended upon for satisfactory results.

MacFarren¹ gives the following as the usual extremes of a well-proportioned flux which has been found very satisfactory:

	Low.	High.
Precipitate.....	100 parts	100 parts
Borax glass.....	12 "	30 "
Sodium carbonate.....	6 "	15 "
Silica.....	3 "	8 "

¹ MacFarren, "Cyanide Practice," 1913, p. 187.

TABLE IV. PROPERTIES OF BULLION-SMELTING FLUXES.

Name.	Symbol.	Chemical nature.	Properties.
Sodium carbonate	Na ₂ CO ₃	Basic	Used extensively; forms sodium silicate with SiO ₂ ; de-sulphurizing agent; forms slags of low freezing point; Na ₂ CO ₃ fuses at 814° C.
Borax or borax glass	Na ₂ B ₄ O ₇ · 10 H ₂ O Na ₂ B ₄ O ₇	Acid	Unites with basic fluxes to form borates, or dissolves metallic oxides and holds them in slag solution; melts at 560° C. Assists in quick fusion and forms fluid slags.
Silica	SiO ₂	Acid	Combines with basic flux to form silicates. Is generally used in combination with borax to give fluid slag. Seldom used. Its chief value is to give fluidity to the slag.
Fluorspar	CaF ₂	Neutral	Melts at 339° C. Used chiefly to oxidize base metals, that they may more readily enter slag, or be removed from bullion as dissolved metallic oxides.
Nitre	KNO ₃	Neutral	More desirable than nitre; higher in available oxygen; crude product cheaply obtained from Chile. Forms more fusible sulphate with sulphur. Freight costs less as equal bulk is more effective than KNO ₃ .
Sodium nitrate	NaNO ₃	Basic	Oxidizing agent, not so destructive to crucible as KNO ₃ or NaNO ₃ . Better oxidizer of lead; not used with silver bullion as it induces silver to enter slag.
Manganese dioxide	MnO ₂	Basic	

From these proportions depending upon the amount of zinc and other base metal present, the flux can be increased or diminished to obtain the best conditions.

Formation of Matte in Smelting. — Matte is a combination of sulphides of base metals; it will always be formed artificially if sulphur and base metals are in the charge. Matte is sometimes formed intentionally to increase the grade of bullion, by extracting base metals from it, due to their affinity for sulphur. It will also diminish the gold and silver contents of a slag, being a good collector of precious metals, but it also prevents some gold and silver from entering the bullion. In general, its formation is undesirable, and may be partly or wholly prevented by roasting precipitate after acid treatment, or by the use of sufficient nitre or other oxidizing agent. An excess of soda will reduce the amount of matte formed, for soda is a desulphurizer, and the basic slag will dissolve the matte and hold it in suspension.

PRECIPITATE TREATMENT PRACTICE.

Smelting and refining the precipitate from cyanide plants is probably one of the simplest processes, in theory, in the whole range of cyanide practice, but until recently it has been the most imperfectly developed detail of them all. This is particularly true of silver producing mills, in which the large quantity of material to be handled at each ~~clean-up~~ is an important factor.

The question as to whether or not acid treatment

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ment should be adopted, depends a great deal upon the ultimate requirements. Where a high-grade product is desired for sale directly to the United States mint, especially when comparatively small quantities of material are to be handled, as with gold ores, there is reason for resorting to acid treatment. An exceptionally high-grade bullion can thereby be produced, which avoids some of the charges for outside refining.

Homestake System of Refining.¹—At the Homestake mills in South Dakota, acid treatment is practiced, bullion averaging 980 fine in gold and silver being produced. The clean-up is made once a month. Zinc dust is the precipitant. The press is blown with air for about two hours before opening it, and the cake is dropped into a tray and carried to the acid treatment department after being sampled and weighed. A lead-lined tank is used for acid treatments, and is provided with agitation mechanism for stirring and mixing the charge. A hood and exhaust fan are provided for carrying away the fumes. (See Figs. 27 and 28.)

Water is run into the treatment tank, the precipitate is added while the agitator is moving, and then acid is introduced slowly, keeping the action under control at all times. Treatment is continued until the action is practically com-

¹ Clark and Sharwood, "Metallurgy of the Homestake Ore," *Bull.* 98, I. M. & M., Nov., 1912, and "Notes on Homestake Metallurgy," by Allan J. Clark, *Bull.* A.I.M.E., July, 1915.

HOMESTAKE SYSTEM OF REFINING 187

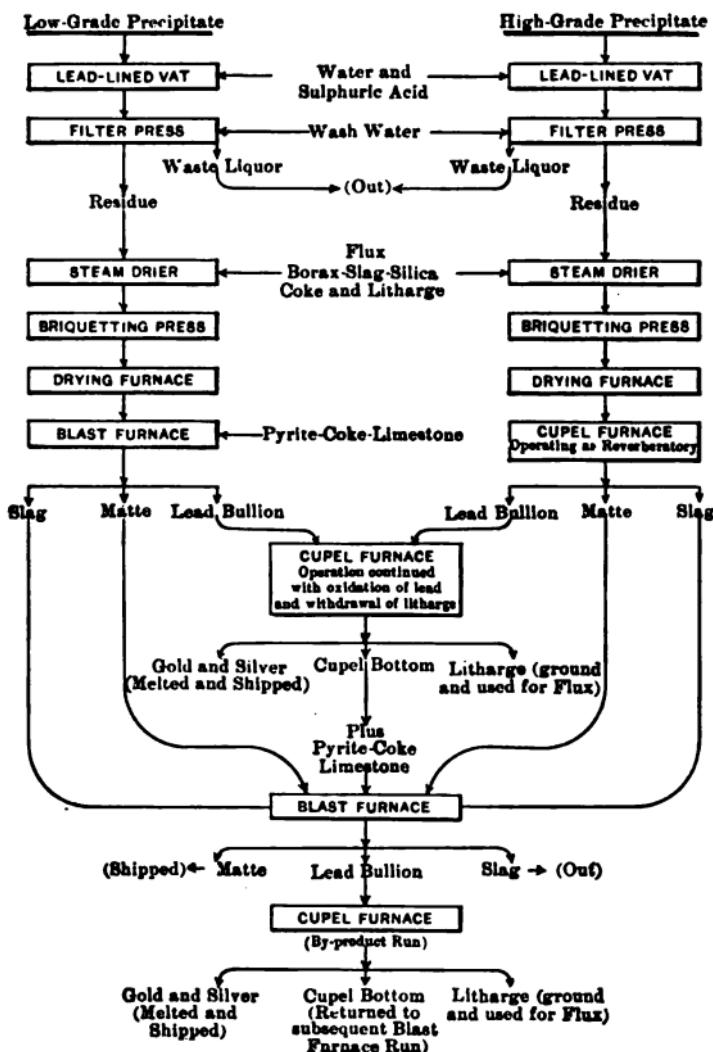


FIG. 28. Homestake System of Precipitate Treatment.

plete, a point determined by testing with methyl orange. The tank is then left to settle, the clear liquor is drawn off, two or three water washes are applied and decanted, and then the whole mass is forced through a filter press and washed again.

The precipitate is now partly dried, reducing its moisture content to about 17 per cent, a consistency which avoids dusting and also assists in forming a sound briquette. The required flux

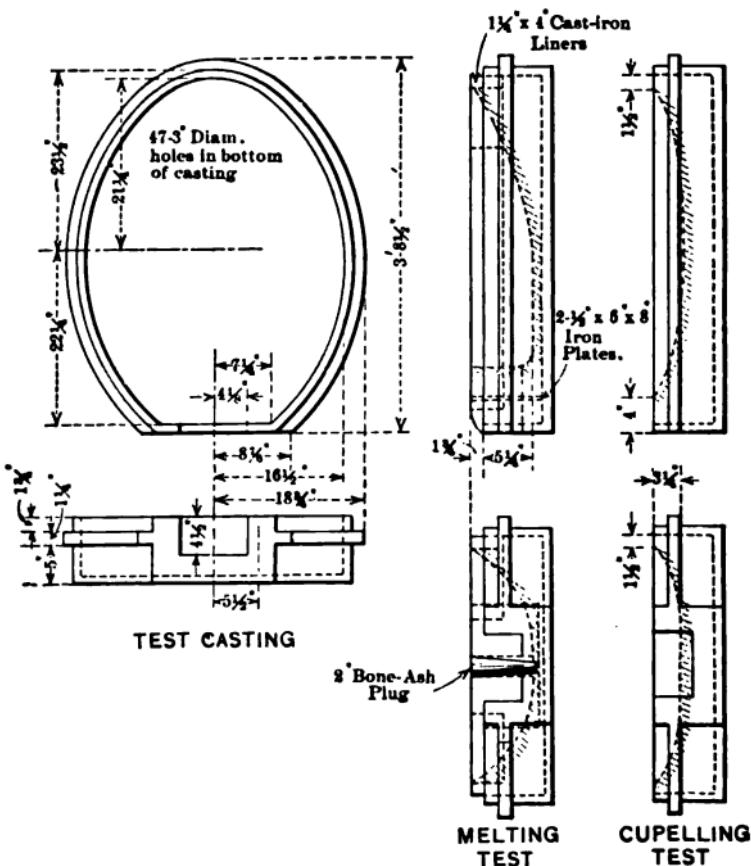


FIG. 29. Furnace Bottoms for Refining Precipitate at Cerro Prieto.

is then added, the ingredients having previously been thoroughly mixed. The homogeneous mixture is then briquetted, each briquette being formed under a pressure of 3500 lb. per sq. in. The briquettes are next dried in a furnace in which

the fire is maintained so low that the pan is not heated to redness directly above the fire box. The briquettes are now sufficiently dry so that they do not spatter when added to the lead bath in the cupel.

This scheme provides for direct cupellation of the briquettes, comprising the high-grade portion of the precipitate; the lower-grade material, which might make a pasty slag in the cupel, is treated in a blast furnace together with other materials, as indicated in Fig. 29.

Direct Smelting of Precipitate. — While acid treatment of precipitate has some definite advantages, by avoiding that operation the actual cost of handling precipitate can be reduced. Thus, for handling large quantities of precipitate, as in large silver producing plants treating high-grade ore, it may be advantageous to omit acid treatment, and reduce the entire precipitate at once to bullion, at the same time getting rid of the greatest possible amount of adulteration by slagging it off during the smelting process. This system is widely followed at the present time, and acid treatment is not so popular as it was. The process of roasting precipitate to volatilize zinc, widely followed at one time, has now been almost universally discarded, as it was shown that an appreciable quantity of gold and silver was unavoidably volatilized along with the zinc.

The operation of melting precipitate into bullion, whether acid treated or not, has been done in many ways, but the prevailing method, until recently, was to melt it, together with the re-

quired flux, in graphite crucibles in a coke-fired wind furnace. In a large or even medium-sized plant, there was usually a battery of these furnaces, varying in number to insure prompt handling of the clean-up. Their manipulation has always been the most severely uncomfortable piece of work connected with the cyanide process. During the last few years, several important improvements have been made which have not only resulted in notable economy, but have brought relief to the workers who attend the process.

Taverner Reverberatory-furnace System. — One of the earliest improvements was made by P. S. Taverner,¹ in dispensing with acid treatment; later this step was resumed as giving a cleaner bullion. The process consists of lead smelting of the precipitate in a reverberatory furnace and cupellation of the lead bullion. This produces a fine doré bullion, which is subsequently re-melted into bars of the required size. The smelting mixture is as follows:

Precipitate.....	100 parts
Litharge.....	100 "
Assay slag.....	55 "
Carbon (coal dust).....	10 "
Silica.....	25 "
Iron (any junk)	

The partly dried charge is shoveled into the furnace and the temperature raised gradually until the moisture has been driven out. When the charge is fused, it is rabbled, and the temperature

¹ Rand Metallurgical Practice, Vol. 1, p. 274.

raised until the slag is fluid. Washes of litharge, carbon, and iron are given to clean the slag, which is usually drawn off through the slag door. After the slag has been removed, the lead is drawn off through a tap hole and run into appropriate molds. Cupellation then follows.

Smelting and Cupellation in One Furnace. — An installation described by H. S. Monroe¹ used a process closely resembling the Taverner system, the only difference being that the same furnace was employed for making lead bullion and cupelling it, using a different bottom for each operation. The precipitate was partly dried over a wood fire, reducing the moisture to about 10 per cent. It was then fluxed, the mixture containing 1 to 100 parts of precipitate, 90 to 120 parts of litharge, 30 parts of old slag, 10 to 20 parts of borax glass, 20 to 30 parts of old assay slag, 8 to 15 parts of silica, depending upon the zinc content of the precipitate, and 5 parts of soda. Coke equal to 5 to 8 parts was sufficient to reduce the litharge. Fig. 29 shows the essential details. The materials for the melting bottom were: 180 lb. limestone, ground through a 20-mesh screen; 180 lb. fireclay; 60 lb. portland cement, and 60 lb. of old test bottoms ground through a 20-mesh screen. For the cupel, the materials were: 156 lb. limestone, ground through 20-mesh; 156 lb. fireclay; 52 lb. portland cement, and 52 lb. old test bottoms, ground through 20-mesh.

¹ "Smelting Precipitate at Cerro Prieto," *Eng. & Min. Jour.*, June 7, 1913.

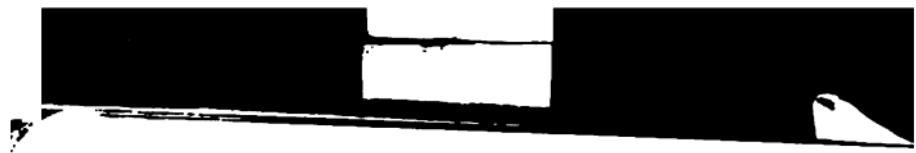
Direct Blast-furnace Treatment at Goldfield.

— Direct blast-furnace smelting of precipitate is practiced at the Goldfield Consolidated mill. The precipitate contains copper, sometimes as much as 40 per cent, but it is satisfactorily handled in the furnace. The mixture consists of precipitate 100 parts; litharge, 100 to 125 parts; blanket concentrates, 60 to 70 parts; and flue dust and floor sweepings, which reduce the moisture to 9 per cent. The mixture is then briquetted, dried, and sent to the blast-furnace. The furnace is about the same as that used at the Homestake and the lead bullion produced is cupelled in about the same way. The product of the cupel is melted in a Steele-Harvey tilting furnace and molded into bars averaging 930 fine.

Fig. 30 shows plan and elevation of the Goldfield refinery, which is particularly well designed. The precipitate is cleaned from presses directly into a steam-jacketed precipitate car, in which the drying is performed. Flux is added and mixed in the same car, which is then moved to the briquetting machine. Briquettes are received on trays and placed in steam-jacketed drying cars. Everything is then conveniently fed into the blast-furnace, thence to the cupel, and finally to the Steele-Harvey furnace, in which the metal is melted for casting into bars.

Oil Fuel and Tilting Furnaces. — Important improvements in the process of melting precipitates were made possible by the introduction of oil as a fuel. The first essential advance was embodied in the well-known Steele-Harvey or

from 1 oz. to 50 lb. per sq. inch.
other similar furnace is the Case, which is
under about the same conditions.



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Monarch furnace, Fig. 31. In this device almost all of the heavy and severe work, which formerly was required in melting, is avoided, and the process is much quicker and cheaper. A point in its favor is the fact that the crucible is not re-



FIG. 31. Steele-Harvey or Monarch Furnace.

moved from the furnace during pouring; by thus avoiding sudden and great changes of temperature, the life of the crucible is much lengthened. This furnace may be used with air pressures varying from 1 oz. to 30 lb. per sq. inch. Another similar furnace is the Case, which is used under about the same conditions.

A later development of the oil-fired tilting furnace is the Monarch-Rockwell, Fig. 32. This is a double-chambered furnace without crucible, the principle being practically that of a reverberatory. The heating is done with an oil burner through a hollow trunnion, upon which the furnace turns for discharging the melted product.

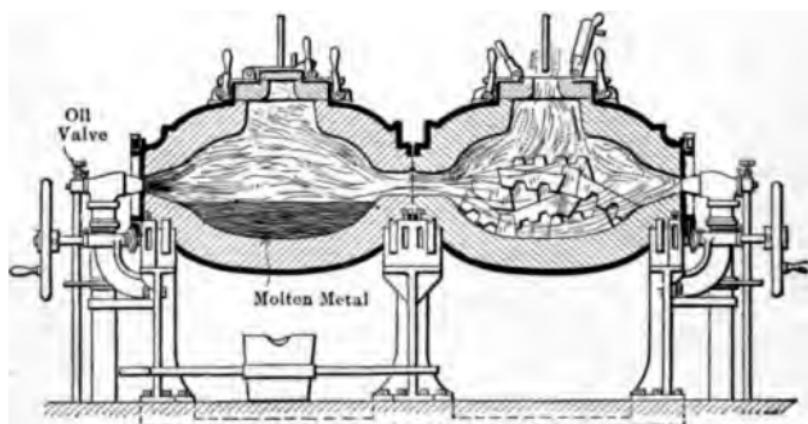


FIG. 32. Monarch-Rockwell Furnace.

The machine is convenient and economical in operation. Low-pressure burners, for gas or oil, are used, the required pressure being from 8 to 12 oz. per sq. inch.

Electricity for Melting Slime. — The use of electricity for smelting precipitate has not been very popular as yet; however, a few types of electric furnace for this purpose have given remarkably successful results. One operated at the Lluvia de Oro mill, Chihuahua, Mexico, is described in detail by H. R. Conklin¹ in a comprehensive review of the furnace and its charac-

¹ *Eng. & Min. Jour.*, June 15, 1912.

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istics. The electric device has been patented and is being marketed by the Pittsburgh Electric Furnace Company.

The furnace consists of a brick-lined steel shell with a stationary electrode at bottom and a movable one at top. Heating is started with an arc, after which the slag or slag-forming constituents, broken glass or soda, borax, and silica, varying proportions, are fed into the furnace. When the slag melts, the electrodes are drawn apart until the desired height of slag is obtained. The furnaces are always worked with a deep slag column and, consequently, a high voltage, these two features tending toward a high power factor and close temperature regulation. There is no need after the first slag is melted, as the upper electrode dips beneath the surface of the melt. The slag column is about half the height of the furnace; after it is formed, the precipitate is added, and sinking to the bottom through the slag, it melts away from the influence of the air.

Résumé of Precipitate Treatment. — Considering the refining process from all sides, it is evident that crucible melting in stationary furnaces is the least desirable method, on account of its inconvenience and its cost. Tilting furnaces are much better; they impose no great hardship upon the operator, and are suitable for the smaller plants, while for a large plant their operating costs are too high. For large plants, the available methods are the reverberatory furnace and cupel, blast-furnace and cupel, cupel alone, and the Monarch-Rockwell direct

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melting system. Experience leads to the conclusion that the Merrill combined blast-furnace and cupel system and the Monarch-Rockwell direct-melting system represent most advanced practice and are most likely to be efficient. It is, of course, to be understood that there are limitations to both, and that each is capable of further development.

Cost of Melting and Refining. — At the Homestake mine the costs of melting and refining are:

Per pound of precipitate.....	\$0.0889
Per ounce of gold.....	0.0953
Per ounce of gold and silver.....	0.0697 ¹
Per ton of solution precipitated.....	0.0085
Per ton of ore treated.....	0.0046

At the Belmont² mill, at Tonopah, Nev., a melt of 105,676 oz. was finished in 36 hr. from the time of lighting the furnace; costs were as follows:

Fuel oil, 1139 gal. @ 3.83¢.....	43.62
Labor.....	40.95
Power for blower.....	6.00
 Total.....	 \$90.57

At the mill of the West End Consolidated, at Tonopah, Nev., the precipitate is melted in a Faber du Faur furnace, Fig. 33, with a special

¹ This figure, which represents the cost per ounce of doré bullion produced, is the one which is most significant, and possibly the best unit for approximate comparison.

² *A. H. Jones, Eng. & Min. Jour.*, June 14, 1913.

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low-pressure burner. The costs here shown were based on about 2 tons of precipitate, which ran about 65 per cent bullion:

	Per oz. of bullion.
Oil fuel.....	\$0.0008
Flux.....	0.0030
Labor.....	0.0008
Crucibles.....	0.0012
Power and extra material.....	0.0004
 Total.....	 \$0.0062



FIG. 33. Faber du Faur Furnace.

The cost per ounce of fine gold-silver bullion at Cerro Prieto, Mexico, using oil as fuel for melting and cupelling, is as follows:

Fuel.....	\$0.006
Flux.....	0.017
Tests.....	0.003
Repairs.....	0.002
Labor.....	0.016
Power.....	0.003
Assaying.....	0.002
 Total.....	 \$0.049

CHAPTER XIII.

CONTROL OF ORE SLIMES.

Definition of Slime. — Slimes are usually defined as all material passing a certain sized sieve, which is generally the finest sieve employed by the metallurgist in making sizing tests; 100-mesh (0.0058 in.) and 200-mesh (0.0029 in.) are the usual limits, and 150-mesh (0.0041 in.) is common.

A reader of the first two volumes of Richard's "Ore Dressing" is impressed by the inflexibility with which all grinding arrangements are praised or condemned according as they produce little or much slime. Until recently, one of the aims of the millman was to keep down the slimes. He was obliged to have separate apparatus for sand and slime treatment. Often it was found more economical to omit treatment of the slimes entirely, or to store them in an immense basin or in heaps, waiting for possible future improvements in methods of treatment.

Recently, improved processes of agitation and filtration have made possible the successful extraction of values from many slimes. The tendency, where the metal is exceedingly finely disseminated, is to omit all sand treatment and to slime everything. Improvements in these processes have been almost entirely of a mechanical

nature, but chemical considerations should not be overlooked as certain reagents may greatly hinder or assist their successful operation.

Slimes and Colloids. — The definition of "slime" as used in ore-dressing is given above. In other words, a slime is a suspension of very fine ore particles in a solution. As for a colloid, numerous definitions can be found. The late Dr. Ashley¹ says: "By the term 'colloid' is meant everything not crystalline. It includes everything covered by the term amorphous. In clays and soils the crystal grains are enveloped with a coating of gelatinous colloidal matter. The same is true of slimes." DeKalb says:² "Every grain will be found enveloped by a film of colloid in addition to that which may be present as free colloid, either as gel or sol, in the solution." Caetani³ says: "A colloid is an amorphous substance of gelatinous nature having a well-marked chemical affinity." And again, "The colloid of one ore will generally differ substantially from the colloid of another ore."⁴ Hayden⁵ says: "The colloids are hydrated amorphous substances resulting from the decomposition of some of the ore constituents, as kaolins, feldspars, etc."

Many other definitions could be cited, but an argument over this matter is quite analogous

¹ "The Chemical Control of Slimes," *Trans. A.I.M.E.*, Vol. XLI, p. 380.

² *Min. & Sci. Press*, Vol. C, 1910, p. 57.

³ *Ibid.*, Vol. CVI, 1913, p. 48.

⁴ *Loc. cit.*, p. 439.

⁵ *Trans. A.I.M.E.*, Vol. XLVI, 1913, p. 243.

to a discussion over the precise meaning of the word "slime," and no ground is gained. What concerns us are certain properties of ores and of intermediate metallurgical products. These properties chance to have been called colloidal, but it is the properties themselves that really concern us. It is usually found that a metallurgist thinks most about colloids in connection with refractory ores, in that slimes produced from them settle slowly or not at all, and are handled with difficulty by the usual metallurgical apparatus. E. E. Free¹ after considerable discussion and criticism of the ordinary conception of colloids, as brought out in the above definition, makes the following statement: "I am not aware that any identifiable colloid differing in any particular from ordinary mineral particles has ever been extracted from such a refractory ore. Indeed, analyses by Hayden of entire slimes at Anaconda, and of colloidal matter washed therefrom, showed very similar compositions, the only important differences being that the colloidal matter was noticeably lower in sulphur and in metals, slightly lower in silica and slightly higher in alumina than the entire slime." In treating slimes, then, we are simply dealing with fine suspension of mineral particles in an aqueous medium. For a complete discussion in this connection the reader is referred to the articles by E. E. Free and O. C. Ralston on "Colloids in Ore Dressing," *Eng. & Min. Jour.*, Feb. 5, 1916;

¹ "Colloids and Colloidal Slimes," *Eng. & Min. Jour.*, Feb. 5, 1916.

March 4, 1916; March 18, 1916; April 15, 1916; April 29, 1916; May 20, 1916; June 3, 1916.

Whether slimes are the same as colloids is non-important, but the chemistry of colloids with respect to electrical charge, adsorptive powers, etc., has an important bearing on the control of slimes. Some ores are colloidal to a high degree, while others, ground under the same conditions, will be more leachable and present little difficulty in settling. Many decomposed ores containing kaolin, feldspar, or hydrated silica have the properties of colloids and require special methods, or chemicals, to produce rapid settling.

Testing Slime Settlement. — No definite rule can be applied to the use of chemicals for increasing the rate of slime settlement. The only procedure is to carry out a series of experiments with the various salts and acids, noting their effect upon the rate of settling. Two ores, when ground to pass a 100-mesh screen, and containing the same percentage of slimes (material passing through 200-mesh), may not settle at the same rate. Each ore must therefore be tested individually; neither does a chemical analysis throw any light upon this property.

For these tests it is well to take a series of graduated cylinders, as in Fig. 34, placing equal amounts of the slime, say 5 to 10 gr., in each cylinder, then adding the desired amount of the coagulant. This chemical should be cheap and one that will not affect the metallurgical process; lime is commonly used. Frequently the most expensive electrolyte is more economical than a



FIG. 34. Cylinders for Determining Settling Rate of Slimes.

ess expensive one. Now dilute with water to a uniform mark, shake, and allow to settle, noting the settling rate. A series of 10 to 15 cylinders can be used, varying the amount of coagulant from the proportion of 1 lb. to 10 or 15 lb. per ton of ore. With lime or other alkali it will generally be noted that the rate of settling will increase up to a certain percentage of coagulant, beyond which the rate will be retarded.

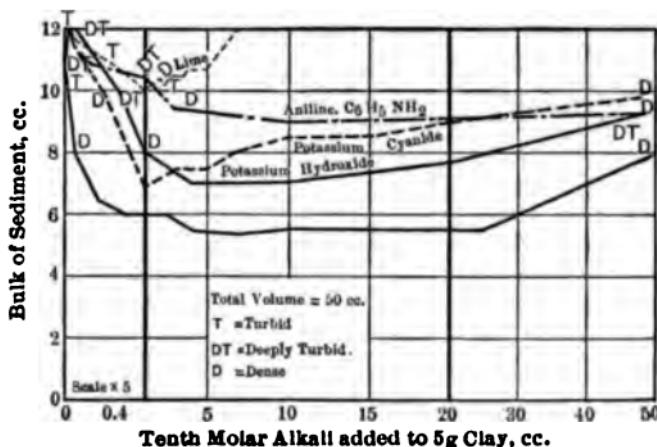


FIG. 35. Effect of Alkalies on Settling of Kaolin.

The effect of several alkalies upon a sample of clay is shown in Fig. 35.¹ In each test 48 c.c. of solution and 5 gr. of clay were put into a 50-c.c. graduated tube, shaken gently for an hour, and allowed to stand over night. The volume occupied by the sediment was measured, and the character of the liquid noted as clear, C; nearly clear, NC; turbid, T; deeply turbid, DT (objects not visible); or dense (light not transmitted).

¹ "The Chemical Control of Slimes," H. E. Ashley, *Trans. A.I.M.E.*, Vol. XLI, p. 380.

A certain small addition of lime acts to put the clay colloid into suspension, a larger amount to coagulate it. Potassium cyanide acts much like an alkali. Fig. 36 shows the effect of cyanide and lime on the same clay. Cyanide in the proportion of 2 lb. per ton has a maximum effect,

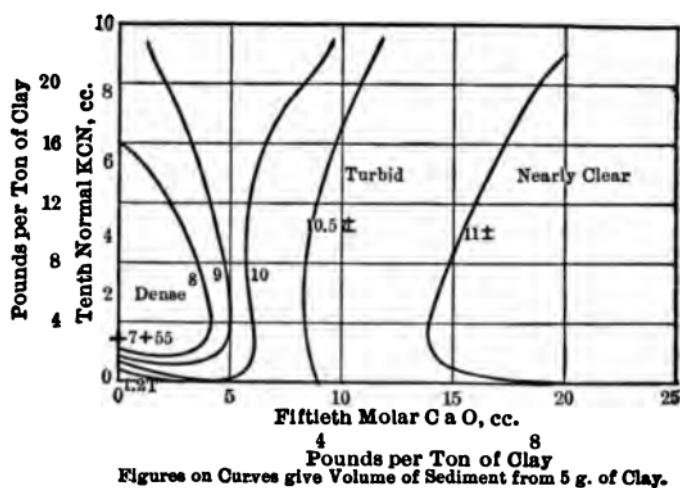


FIG. 36. Effect of Cyanide and Lime on Settling of Kaolin.

reducing the bulk of the sediment from 12 c.c. to 7 c.c. The addition of about 0.7 lb. of lime (per ton) still further reduces the sediment to 5.5 c.c. The sediment is entirely crystalline, all of the colloid matter being in suspension. If the lime is increased to 6.5 lb. per ton, the liquid becomes nearly clear by coagulation of the colloid matter.

It has been shown independently by Comstock¹

¹ *Electrochemical & Metallurgical Industry*, Vol. VII, No. 2, p. 74.

and Ashley¹ that the rate of settling of a slime, so far as temperature is concerned, varies inversely as the viscosity of the water. A rise of temperature decreases the viscosity of water (Table V), and a quicker settling of slime is the result.

TABLE V. VISCOSITY OF WATER.¹

Temperature.		Viscosity C.G.S. units.	Percentage of decrease of viscosity per 10° C.
Deg. C.	Deg. F.		
0	32	0.01793
10	50	0.01311	26.88
20	68	0.01006	23.26
30	86	0.00800	20.48
40	104	0.00657	17.87
50	122	0.00550	16.29
60	140	0.00469	14.73
70	158	0.00406	13.43
80	176	0.00356	12.31
90	194	0.00316	11.24
100	212	0.00284	10.13

¹ Hosking, *Proc. Roy. Soc. New South Wales*, Vol. XCIII, pp. 34-38 (1910).

The most important practical deduction to be drawn from this table is that the effect of temperature change upon viscosity, and hence upon rate of settling, is greater at low temperatures than at high. For instance, a rise of temperature from freezing to 20° C. (68° F.) diminishes viscosity nearly 44 per cent and will probably increase the rate of settling by nearly as much. Therefore, there may be cases in which the use of waste steam, to prevent chilling of slimes in

¹ *Min. & Sci. Press*, Vol. XCVIII, No. 24, p. 831 (June, 1909).

the winter, would prove of important practical value, although heating above normal summer temperature might be unprofitable.

Acceleration of settling by increased flocculation is not always to be desired, as the floccules formed by the particles have no great mechanical rigidity, but persist to some extent in the settled material with the result that a well-flocculated slime, when thickened, may have a large pore space and greater volume per unit of dry weight than a slime that settled with a lower degree of flocculation. In cases where the volume of a settled slime and the retention of solution in it must be minimized, the engineer must strike a balance between low flocculation, meaning slow settling with good consolidation, and high flocculation, meaning rapid settling with less perfect consolidation. The degree of consolidation caused by an addition agent will generally be different from that caused by increasing temperature; the latter has no effect upon degree of flocculation.

If it is desired to settle out granular matter, as in the classification of sands and slimes, the most favorable proportion of alkali to add is that amount which causes full deflocculation. The action of an alkali in dissolving the colloid matter from the surface of crystal grains breaks up the floccules, or lumps, and is called "deflocculation" by agricultural chemists. The manner in which a salt of a bivalent metal suppresses the deflocculating effect of an alkali is shown in Fig. 37, the additions of sodium hydroxide being repre-

sented by abscissas. Thus, with no NaOH, the clay was partially deflocculated; with about 7 c.c. of N/10 NaOH (point P) the clay was fully deflocculated; and further additions up to a total of 85 c.c. of N/10 NaOH (point C) continued to keep all the colloid in suppression. The

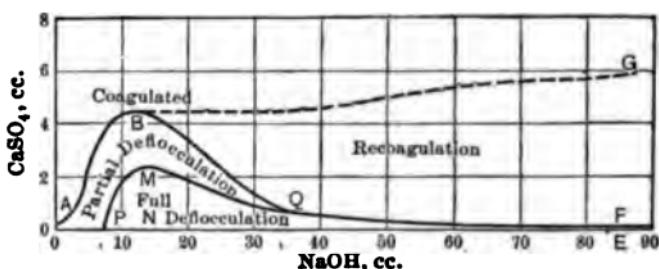


FIG. 37. Opposition of Calcium Sulphate to Deflocculation by Sodium Hydroxide.

additions of calcium sulphate are represented by ordinates. About 0.2 c.c. of $\frac{1}{10}$ molar¹ CaSO_4 (point A) sufficed completely to coagulate the clay, with complete clarification of the liquid; and further additions up to a total of 8 c.c. of $\frac{1}{10}$ molar CaSO_4 continued to keep the clay wholly coagulated. When both calcium sulphate and sodium hydroxide were added to the clay, the colloid matter was wholly dissolved only with those quantities represented by the points in the area *PMQFEP*. In other words, with this clay, slight additions of CaSO_4 very rapidly narrowed the range of concentrations of NaOH effecting deflocculation, until 2 c.c. $\frac{1}{10}$ molar CaSO_4 limited

¹ $\frac{1}{10}$ of molecular weight of substance dissolved in 1000 c.c. H_2O .

full deflocculation to but one proportion of N/10 NaOH, viz., 15 c.c., at point *M*. Concentrations of calcium sulphate given by the line *ABG* kept the clay wholly coagulated, and at nearly the original bulk. Larger amounts of calcium sulphate also kept the clay coagulated, but the bulk became gradually larger as the distance above the line *ABG* increased. In the area *QBGF*, the clay was momentarily deflocculated as if by only 15 c.c. N/10 NaOH when the sodium hydroxide was first added; but after thorough mixture and complete reaction the clay was coagulated.

If it is desired to transport mixed colloids and granular matter in a launder, the grains will be carried by the least amount of water, or least slope, when the condition of the mixture is represented by a portion in the area marked "recoagulation."

It has been found that pure water has a decided solvent effect upon crystalline minerals, increasing with temperature. Crushing in cold water will therefore minimize this solvent effect. Thus, silica dissolved in water may be "salted out" by the impurities in the quartz, giving rise to colloidal silica.

A slime that is partly in the "sol" state¹ neither settles nor filters well; it must be fully coagulated. If an excess of coagulant is used the viscosity of the solution is raised and the rate of settling retarded.

¹ It is customary to call the colloid a "gel" in its solid condition and a "sol" when in solution or suspension.



TESTING SLIME SETTLEMENT 209

An excellent article by H. S. Coe and G. H. Clevenger, "Laboratory Method for Determining the Capacities of Slime-settling Tanks," was published in *Bulletin 111* of the A.I.M.E., Mar., 1916.

CHAPTER XIV.

CYANIDE TREATMENT OF FLOTATION CONCENTRATE.

The difficulties attending the treatment of concentrate by cyanide are well known. The process of concentration collects in a small bulk not only the valuable constituents of the ore but also those substances which act as cyanicides, or are readily converted by oxidation or otherwise into cyanicides, so that their effect, per ton of material treated, is greater than would be the case with unconcentrated ore. Heavy minerals, such as the sulphides of iron, copper, lead, arsenic, antimony, zinc, and double sulphides such as mispickel, proustite, pyrargyrite, and bornite, naturally tend to accumulate in the concentrate. If some time elapses between the collection of this concentrate and its treatment, oxidation may take place, with the formation of sulphates, antimonates, and arsenates, which are still more detrimental to cyanide treatment than the original minerals.

The value of the cyanide process for the treatment of a flotation concentrate rests on two points, namely, the grade of the tailing produced and the net proceeds from the operation. The latter point is complicated by questions of geographical situation, for if the concentrate can-

not be treated locally the cost of realization may be so heavy that flotation would be entirely precluded.

Methods of Treatment. — The ordinary direct cyanide process is of little value and the following treatments may or may not give satisfactory results.

1. Preliminary water, acid, or alkali wash.
2. Preliminary roasting.
3. Fine grinding.
4. Special solvents, such as bromocyanide.
5. Prolonged contact of the material with cyanide, for periods of several months.

Interference of Oil. — Besides the usual difficulties attending the cyanidation of ordinary products of gravity concentration, flotation concentrate will give additional trouble, due to the presence of oil, tar, or other flotation agent. As the particles of sulphides will be coated with a film of oil which will prevent the solution from coming in contact with them, some special treatment will be required to remove this film. Oil may cause the same trouble as that produced by carbonaceous matter, the re-precipitation of values dissolved in the cyanide solution.

For results of investigations along this line the reader is referred to the following articles:

“The Cyanide Treatment of Flotation Concentrate,” by C. Butters and J. E. Clennell. *Min. & Sci. Press*, Nov. 20, 1915.

“Cyanidation of Flotation Concentrates,” by P. W. Avery. *Min. & Sci. Press*, May 6, 1916.

“Cyaniding Flotation Concentrates,” by H. R. Laying. *Min. & Sci. Press*, June 3, 1916.

"Cyanidation of Flotation Concentrates," by P. W. Avery. *Min. & Sci. Press*, May, 1916.

"Cyanide Treatment of Flotation Concentrates." *Min. & Sci. Press*, Nov. 20, 1915.

"Solution Control in Cyanidation," by A. W. Allen. *Min. & Sci. Press*, Jan. 30, 1915, p. 186.

"Flotation and Cyanidation." *Met. & Chem. Eng.*, May 15, 1916.

CHAPTER XV.

FIRST AID IN CYANIDE POISONING.

Considering the large quantities of cyanide salt and solution handled in gold and silver mining districts, and the poisonous nature of this useful compound, the number of deaths attributable to it is small. When a man has inhaled prussic acid gas, or has swallowed some cyanide solution, the poison acts very quickly, and his life depends on immediate action. The gas, when pure, causes almost instant death; and when diluted with air, it causes dizziness, faintness, and a depressing headache. Solutions also act quickly internally, while with many men they act on the skin and produce painful eruptions.

Cyanogen Gas Poisoning. — If an employee has inhaled prussic acid gas, proceed as follows:¹ Dash water on the patient's face; start artificial respiration; make him inhale either a small quantity of ammonia, ether, or chlorine gas. The latter may be quickly made by sprinkling a small amount of chloride of lime on a flannel cloth moistened with acetic acid, and then holding the cloth to the nostrils of the patient.

Cyanide Solution Poisoning. — Place the patient in a hot bath, if procurable, and apply cold

¹ "First Aid for Cyanide Poisoning," by W. K. Kritzer. *Min. & Sci. Press*, Jan. 18, 1913.

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water to the spine and neck, provided that no delays are permitted to intervene in carrying out previous instructions. Also excite vomiting by tickling the back of the patient's throat with a finger or feather; by giving luke-warm water, or strong mustard and water; by using a stomach tube and hot water; or by physical means. Diluted solutions of ammonia, cobalt nitrate, peroxide of hydrogen, or freshly precipitated carbonate of iron may be given. The last mentioned is made by mixing equal parts of ferrous sulphate, and sodium carbonate, and then administering at once.

Emergency Kit. — It is advisable to keep on hand an outfit comprising one sealed bottle containing 30 c.c. of caustic potash; one sealed bottle containing 30 c.c. of 33 per cent solution of ferrous sulphate; and one sealed package containing oxide of magnesium. As quickly as possible, empty the contents of the two bottles and the package into a metal cup, and stir thoroughly with a metal spoon. If the patient is conscious, make him swallow the mixture at once and then lie down a few minutes. If unconscious, place him on his back and pour the mixture down the throat in small quantities, if necessary pinching the nose to start swallowing. Then incite vomiting by one of the methods suggested above. It is advisable to have a soft rubber stomach-tube having a funnel and an exhaust bulb.

First-aid Cabinet. — In all cyanide plants it is important to have wooden cabinets, with com-

partments of suitable size, in conspicuous and easily accessible parts of the building, preferably painted red. Fasten the emergency orders for the handling of patients to the inside of the cabinet door, properly label each bottle, and see that the contents are kept in a fresh and pure condition. The four compartments should contain: (1) One large bottle of distilled water; one large metal spoon; and one pint metal cup for mixing the different antidotes. (2) For external use, one bottle each of ammonia, ether, acetic acid, chloride of lime; and a piece of flannel cloth for administering the chlorine gas. (3) For internal use, one bottle each of diluted ammonia, cobalt nitrate, ferrous sulphate, and sodium carbonate, the two latter to be fresh. (4) One sealed bottle with 30 c.c. of caustic potash, one sealed bottle with 30 c.c. of 33-per cent solution of ferrous sulphate, and one sealed package of oxide of magnesium. An excellent preventative measure, now fortunately observed at most cyanide plants, is the erection of notice-boards in different parts, giving warning that certain tanks and pipes contain cyanide solution and others water.

CHAPTER XVI.

USEFUL DATA.

ENERGY.

Torque units should be distinguished from energy units: Thus, foot-pound and kilogram-meter for energy, and foot-pound and meter-kilogram for torque.

1 ft.-lb.¹ = 13,560,000 ergs = 1.356 joules = 0.3239 g.-cal. = 0.1383 kg.-m. = 0.001285 B.t.u. = 0.0003766 watt-hr. = 0.0000005051 h.p.-hr.

1 kg.-m. = 98,060,000 ergs = 9.806 joules = 7.233 ft.-lb. = 2.34 g.-cal. = 0.009296 B.t.u. = 0.002724 watt-hr. = 0.000003704 h.p.-hr. (metric).

1 B.t.u. = 1055 joules = 778.1 ft.-lb. = 252 g.-cal. = 107.6 kg.-m. = 0.5555 lb.-centigrade heat unit = 0.2930 watt-hr. = 0.252 kg.-cal. = 0.0003984 h.p.-hr. (metric) = 0.0003930 h.p.-hr.

1 watt-hr. = 3600 joules = 2655.4 ft.-lb. = 860 g.-cal. = 367.1 kg.-m. = 3.413 B.t.u. = 0.001341 h.p.-hr.

1 h.p.-hr. = 2,684,000 joules = 1,980,000 ft.-lb. = 273,700 kg.-cm. = 745.6 watt-hr.

1 kw.-hr. = 2,655,000 ft.-lb. = 367,190 kg.-m. = 1.36 h.p.-hr. (metric) = 1.34 h.p.-hr.

POWER.

1 g.-cm. per sec. = 0.00009806 watt.

1 ft.-lb. per min. = 0.02260 watt = 0.00003072 h.p. (metric) = 0.00000303 h.p.

1 watt = 44.26 ft.-lb. per min. = 6.119 kg.-m. per min. = 0.001 kilowatt.

¹ The hyphen (-) as used here means "multiplied by."

1 h.p. = 33,000 ft.-lb. per min. = 745.6 watts = 550 ft.-lb. per sec. = 76.04 kg.-m. per sec. = 1.01387 h.p. (metric).

1 kw. = 44,256.7 ft.-lb. per min. = 101.979 kg.-m. per sec. = 1.3597 h.p. (metric) = 1.341 h.p.

RESISTIVITY.

1 ohm per cir. mil-ft. = 0.7854 ohm per sq. mil-ft. = 0.001662 ohm per sq. mm.-m. = 0.0000001657 ohm per cu. cm. = 0.0000000524 ohm per cu. in.

1 ohm per sq. mil-ft. = 1.273 ohms per cir. mil-ft. = 0.002117 ohm per sq. mm.-m = 0.0000002116 ohm per cu. cm. = 0.00000008335 ohm per cu. in.

1 ohm per cu. in. = 15,280,000 ohms per cir. mil-ft. = 12,000,000 ohms per sq. mil-ft. = 25,400 ohms per sq. mm.-m. = 2.54 ohms per cu. cm.

CURRENT DENSITY.

1 amp. per sq. in. = 0.7854 amp. per cir. in. = 0.1550 amp. per sq. cm. = 1,273,000 cir. mils per amp. = 0.000001 amp. per sq. mil.

1 amp. per sq. cm. = 6.45 amp. per sq. in. = 197,000 cir. mils per amp.

1000 cir. mils per amp. = 1273 amp. per sq. in.

1000 sq. mils per amp. = 1000 amp. per sq. in.

METRIC SYSTEM WITH CONVERSIONS.

10 milli- = 1 centi-.
10 centi- = 1 deci-.
10 deci- = 1 (unit).
10 (units) = 1 deca-.

10 deca- = 1 hecto-.
10 hecto- = 1 kilo-.
1 kilo- = 1 myria-.

WEIGHT.

METRIC UNIT IS GRAM.

Gram = weight 1 cubic centimeter of water at 4° C.

Gram = 15.4324 grains.

Gram = 0.03215 ounce troy.

Gram = 0.00267923 pound troy.
Gram = 0.03527 ounce avoirdupois.
Gram = 0.00220462 pound avoirdupois.
Milligram = 0.0154324 grain.
Kilo or kilogram = 32.15 ounces troy.
Kilo or kilogram = 2.67923 pounds troy.
Kilo or kilogram = 35.27 ounces avoirdupois.
Kilo or kilogram = 2.20462 pounds avoirdupois.
Metric ton = 1000 kilos or kilograms.
Metric ton = 2204.62 pounds avoirdupois.
Metric ton = 1.10231 United States tons (2000 pounds).
Grain = 0.0648 gram.
Ounce troy = 31.10348 grams.
Pound troy = 0.37324 kilo or kilogram.
Pound troy = 373.24 grams.
Ounce avoirdupois = 28.3495 grams.
Pound avoirdupois = 0.45359 kilo or kilogram
Pound avoirdupois = 453.59 grams.
Ton (2000 pounds) = 0.90718 metric ton.
Ton (2000 pounds) = 907.185 kilograms.
Assay ton = 29.1666 grams.
Assay ton = 0.0377 ounce troy.
Assay ton = 0.07814 pound troy.
Assay ton = 1.0287 ounces avoirdupois.
Assay ton = 0.0643 pound avoirdupois.

CAPACITY.

METRIC UNIT IS LITER.

Liter = 1000 cubic centimeters.
Liter = 0.26417 gallon (231 cubic inches).
Liter = 1.05668 quarts.
Liter = 33.81 liquid ounces.
Liter = 61.023 cubic inches.
Gallon (231 cubic inches) = 3.78543 liters.
Gallon (231 cubic inches) = 3785.43 cubic centimeters.
Liquid ounce = 0.029574.

VOLUME.

Cubic meter = 35.314 cubic feet.
Cubic meter = 1.3079 cubic yards.
Cubic centimeter = 0.061 cubic inch.
Cubic foot = 0.02832 cubic meter.
Cubic yard = 0.7645 cubic meter.

LENGTH.**METRIC UNIT IS METER.**

Meter = 39.37 inches.	Inch = 2.54 centimeters.
Meter = 3.280833 feet.	Foot = 30.48 centimeters.
Kilometer = 3280.833 feet.	Foot = 0.3048 meter.
Kilometer = 0.62137 mile.	Mile = 1.60935 kilometers.
Centimeter = 0.3937 inch.	Mile = 1609.347 meters.

AREA.

Square meter = 10,764 square feet.
Square meter = 1550.3 square inches.
Hectare or square hectometer = 2.4711 acres.
Square kilometer = 247.1 acres.
Square inch = 6.452 square centimeters.
Square foot = 929 square centimeters.
Square foot = 0.0929 square meter.
Square mile = 2.59 square kilometers.
Acre = 0.40469 hectare.
Acre = 4046.9 square meters.

UNITED STATES WEIGHTS AND MEASURES.**AVOIRDUPOIS WEIGHT.**

27.34375 grains = 1 dram.
16 drams = 1 ounce (oz.).
437.5 grains = 1 ounce.
16 ounces = 1 pound (lb.).
7000 grains = 1 pound.
100 pounds = 1 hundredweight.
2000 pounds = 1 short ton (usually used)
2240 pounds = 1 long ton (seldom used).

TROY WEIGHT.

24 grains = 1 pennyweight (dwt.).
 20 pennyweights = 1 ounce (oz.).
 480 grains = 1 ounce.
 12 ounces = 1 pound (lb.).
 5760 grains = 1 pound.

APOTHECARIES' WEIGHT.

20 grains = 1 scruple.	480 grains = 1 ounce.
3 scruples = 1 dram.	12 ounces = 1 pound.
8 drams = 1 ounce.	5670 grains = 1 pound

LENGTH.

12 inches = 1 foot.	4 rods = 1 chain.
3 feet = 1 yard.	66 feet = 1 chain.
5½ yards = 1 rod.	320 rods = 1 mile.
16½ feet = 1 rod.	5280 feet = 1 mile.

AREA.

144 square inches = 1 square foot.
 9 square feet = 1 square yard.
 30½ square yards = 1 square rod.
 160 square rods = 1 acre = 43,560 square feet.
 640 acres = 1 square mile.

VOLUME.

1728 cubic inches = 1 cubic foot.
 27 cubic feet = 1 cubic yard.

CAPACITY.

Liquid.

4 gills = 1 pint.
 2 pints = 1 quart.
 4 quarts = 1 gallon (231 cubic inches).
 63 gallons = 1 hogshead.
 31½ gallons = 1 barrel.

Dry.

2 pints = 1 quart.

4 quarts = 1 gallon (268.8025 cubic inches).

2 gallons = 1 peck.

4 pecks = 1 bushel (2150.42 cubic inches).

AVOIRDUPOIS AND TROY CONVERSIONS.

Ounce troy = 1.09714 ounces avoirdupois.

Pound troy = 13.166 ounces avoirdupois.

Pound troy = 0.822857 pound avoirdupois.

Ounce avoirdupois = 0.91145 ounce troy.

Pound avoirdupois = 14.583 ounces troy.

Pound avoirdupois = 1.21528 pounds troy.

Ton (2000 pounds avoirdupois) = 29,166 $\frac{1}{3}$ ounces troy.

Ton (2000 pounds avoirdupois) = 2430.56 pounds troy.

MONEY.**ENGLISH.**

4 farthings = 1 penny (d). 1 pound = 7.3224 grams

4 pence = 1 shilling (s). gold.

20 shillings = 1 pound (£). 1 pound = \$4.8665 United

1 pound = 113.001 grains gold. States money.

MEXICAN.

100 centavos = 1 peso. 1 peso = .87 or $\frac{1}{2}$ troy ounce

1 peso = 417.74 grains silver. (approximate) silver.

1 peso = 27.073 grams silver.

UNITED STATES.

100 cents = 1 dollar (\$). 1 dollar = 23.22 grains gold.

VALUE OF GOLD.

1 ounce troy = \$20.67.
 1 pennyweight (dwt.) = $\frac{1}{20}$ ounce troy.
 1 pennyweight = \$1.03 $\frac{7}{10}$.
 1 grain = 4.306 cents (United States).
 1 gram = 0.6646.
 1 gram = 0.03215 or $\frac{1}{31}$ (approximate) ounce troy.
 1 kilo = \$664.60.
 1 kilo = 32.15 ounces troy.

CONVERSION OF THERMOMETER READINGS.

	Freezing point.	Boiling point.
	degrees.	degrees.
Fahrenheit.....	32	212
Centigrade.....	0	100

To convert Fahrenheit to Centigrade, subtract 32 and multiply by $\frac{5}{9}$.

To convert Centigrade to Fahrenheit, multiply by $\frac{9}{5}$ and add 32.

WEIGHT AND MEASURE OF WATER.

1 pound (avoirdupois) water = 27.68122 cubic inches.
 1 pound (avoirdupois) water = 0.0160192 cubic foot.
 1 gallon (United States liquid) water = 231 cubic inches.
 1 gallon (United States liquid) water = 0.13368 cubic foot.
 1 gallon (United States liquid) water = 3.78543 liters.
 1 gallon (United States liquid) water = 8.3389 pounds (avoirdupois).
 1 cubic foot water = 62.42 pounds (avoirdupois).
 1 cubic foot water = 7.48052 gallons.
 1 cubic foot water = 28.318 liters.
 1 ton water = 339.84 gallons.
 1 ton water = 32.941 cubic feet.
 1 ton water = 907.2 liters.
 1 liter water = 2.2046 pounds (avoirdupois).

WEIGHT OF ROCK AND SAND 223

WEIGHT OF ROCK AND SAND.

	Cubic feet per ton.	Weight in pounds per cubic foot.
Sulphide ore in place.....	11-13	182-154
Sulphide ore broken.....	15-18	133-111
Oxidized ore in place.....	14-18	143-111
Oxidized ore broken.....	22-24	91-81
Quartz in place. (Specific gravity, 2.65).....	12	165
Quartz broken.....	21	94
Earth in bank.....	18	111
Earth, dry and loose.....	27	74
Clay.....	17	118
Loose sand.....	25	80
Mill tailing. (Specific gravity, 2.7).....		
Sand collected under water.....	21.5	93
Transferred sand (before leaching).....	26	77
Leached sand (that has been transferred).....	24	83.3

SPECIFIC GRAVITIES OF THE COMMON METALS AND MINERALS.

	Symbol.	Sp. gr.
Aluminum.....	Al	2.6
Amalgam	Ag ₂ Hg ₃	13.75-14.1
Amphibole..... {	(Sec. 1, Table VII, 2, 3)	{ 2.9-3.4
Anglesite.....	PbSO ₄	6.1-6.4
Anthracite.....	1.32-1.7
Antimony.....	Sb	6.5-6.7
Apatite.....	Ca ₅ (Cl, F)(PO ₄) ₃	3.17-3.23
Argentite.....	Ag ₂ S	7.2-7.6
Arsenic.....	As	5.6-5.7
Arsenopyrite.....	FeAsS	6.0-6.2
Azurite.....	Cu ₃ (OH) ₂ (CO ₃) ₂	3.8
Barite.....	BaSO ₄	4.3-4.6
Biotite..... {	(H.K.) ₂ (Mg, Fe) ₂ Al ₂ (SiO ₄) ₃	{ 2.7
Bismuth.....	Bi	9.7-9.8
Bitum. coal.....	1.14-1.4
Bornite.....	Cu ₅ FeS ₄	4.9-5.4
Brown coal.....	1.15-1.3
Calamine.....	(ZnOH) ₂ SiO ₂	4.3-4.5
Calcite.....	CaCO ₃	2.7
Cassiterite.....	SnO ₂	6.8-7.1
Cast iron.....	7.3-7.73
Cerargyrite.....	AgCl	5.0-5.5
Cerussite.....	PbCO ₃	6.5-6.6
Chalcocite.....	Cu ₂ S	5.5-5.8
Chalcopyrite.....	CuFeS ₂	4.1-4.3
Chromite.....	FeCr ₂ O ₄	4.3-4.6
Chromium.....	Cr	6.5
Chrysocolla.....	CuSiO ₃ .2H ₂ O	2.0-2.3
Cinnabar.....	HgS	8.0-8.2
Cobalt.....	Co	8.6
Cobaltite.....	CoAs ₂	6.0-6.1
Copper (native).....	Cu	8.8-8.9
Corundum.....	Al ₂ O ₃	3.9-4.1
Cuprite.....	Cu ₂ O	5.8-6.1
Dolomite.....	CaMg(CO ₃) ₂	2.8-2.9
Enargite.....	Cu ₃ AsS ₄	4.4
Epidote..... {	Ca ₂ (Al, Fe) ₂ (AlOH)(SiO ₄) ₃	{ 3.2-3.5
Fluorite.....	CaF ₂	3.0-3.3
Galenite.....	PbS	7.4-7.6
Garnet.....	R ₃ R ₂ (SiO ₄) ₃	3.1-4.3
Gold (native).....	Au	15.6-19.3
Graphite.....	C	2.1-2.2
Gypsum.....	CaSO ₄ .2H ₂ O	2.3

COMMON METALS AND MINERALS 225

SPECIFIC GRAVITIES OF THE COMMON METALS AND MINERALS (Continued).

	Symbol.	Sp. gr.
	Fe_2O_3	4.9-5.3
	(Au,Ag)Te	8.3-8.6
e	Ir	22.42
e)	(Ir,Os)	19-21
e)	Fe	7.86
	Pb	11.37
	$\text{Fe}_2(\text{OH})_6\text{FeO}_3$	3.6-4.0
	Fe_3O_4	4.9-5.2
	$\text{Cu}_2(\text{OH})_2\text{CO}_3$	3.9-4.0
e	Mn	7.39
	FeS_2	4.6-4.9
	Hg	13.59
	NiS	5.3-5.6
ite	MoS	4.6-4.9
e	(HK)AlSiO ₄	2.8-3.0
	NiAs	7.2-7.7
	Ni	8.9
		0.85-0.9
(native)	Pt	14-19
e	(Ag,Cu) ₂ SbS ₃	6.0-6.2
	Ag ₃ AsS ₃	5.6-5.7
te	Ag ₃ SbS ₃	5.7-5.8
	FeS_2	4.9-5.2
	$\text{CaMg}(\text{SiO}_3)_2$	3.2-3.6
e	FeS	4.5-4.6
	SiO_2	2.64
osite	MnCO_3	3.5-4.5
e	$\text{H}_2\text{Mg}_3\text{SiO}_3\text{O}_2$	2.5-2.6
tive)	FeCO ₃	3.8-3.9
ite	Ag	10.1-11.1
e	ZnCO ₃	4.3-4.5
e	ZnS	3.9-4.1
e	Ag ₃ SbS ₄	6.2-6.3
	Sb_2S_3	4.5-4.6
	S	2.07-2.09
	(Au,Ag)Te ₂	7.9-9.0
	$\text{H}_2\text{Mg}_3(\text{SiO}_3)_4$	2.5-2.9
ite	$\text{Cu}_3\text{Sb}_2\text{S}_7$	4.5-5.1
	Sn	7.29
	CaSiTiO_5	3.4-3.5
e	(FeMn)WO ₄	7.1-7.5
	Zn	7.15
	ZnO	5.4-5.7

INTERNATIONAL ATOMIC WEIGHT.

Element.	Symbol.	Atomic weight.	Element.	Symbol.	Atomic weight.
Aluminum.....	Al	27.1	Molybdenum.....	Mo	96.0
Antimony.....	Sb	120.2	Neodymium.....	Md	144.3
Argon.....	A	39.88	Neon.....	Ne	20.2
Arsenic.....	As	74.96	Nickel.....	Ni	58.68
Barium.....	Ba	137.37	Nitrogen.....	N	14.01
Bismuth.....	Bi	208.0	Osmium.....	Os	190.9
Boron.....	B	11.0	Oxygen.....	O	16.0
Bromine.....	Br	79.92	Palladium.....	Pd	106.7
Cadmium.....	Cd	112.40	Phosphorus.....	P	31.04
Cesium.....	Cs	132.81	Platinum.....	Pt	195.2
Calcium.....	Ca	40.09	Potassium.....	K	39.10
Carbon.....	Ca	12.0	Praseodymium.....	Pr	140.6
Cerium.....	Ce	140.25	Radium.....	Ra	226.4
Chlorine.....	Cl	35.46	Rhodium.....	Rh	102.9
Chromium.....	Cr	52.0	Rubidium.....	Rb	85.45
Cobalt.....	Co	58.97	Ruthenium.....	Ru	101.7
Columbium.....	Cb	93.5	Samarium.....	Sa	150.4
Copper.....	Cu	63.57	Scandium.....	Sc	44.1
Dysprosium.....	Dy	162.5	Selenium.....	Se	79.2
Erbium.....	Er	167.4	Silicon.....	Si	28.3
Europium.....	Eu	152.0	Silver.....	Ag	107.88
Fluorine.....	F	19.0	Sodium.....	Na	23.0
Gadolinium.....	Gd	157.3	Strontium.....	Sr	87.63
Gallium.....	Ga	69.9	Sulphur.....	S	32.07
Germanium.....	Ge	72.5	Tantalum.....	Ta	181.0
Glucinum.....	Gl	9.1	Tellurium.....	Te	127.5
Gold.....	Au	197.2	Terbium.....	Tb	159.2
Helium.....	He	3.99	Thallium.....	Tl	204.0
Hydrogen.....	H	1.008	Thorium.....	Th	232.4
Indium.....	In	114.8	Thulium.....	Tm	168.5
Iodine.....	I	126.92	Tin.....	Sn	119.0
Iridium.....	Ir	193.1	Titanium.....	Ti	48.1
Iron.....	Fe	55.85	Tungsten.....	W	184.0
Krypton.....	Kr	82.92	Uranium.....	U	238.5
Lanthanum.....	La	139.0	Vanadium.....	V	51.06
Lead.....	Pb	207.10	Xenon.....	Xe	130.2
Lithium.....	Li	6.94	Ytterbium.....	Yb	172.0
Lutecium.....	Lu	174.0	Yttrium.....	Yt	89.0
Magnesium.....	Mg	24.32	Zinc.....	Zn	65.37
Manganese.....	Mn	54.93	Zirconium.....	Zr	90.6
Mercury.....	Hg	200.0

MAXIMUM SOLUBILITIES.

(IN WATER AT ORDINARY TEMPERATURES.)

Substance.	Formula.	Solubility.
Aluminum sulphate.....	$\text{Al}_2(\text{SO}_4)_3$	1 part in 3
Calcium carbonate.....	CaCO_3	Insoluble
Calcium chloride.....	CaCl_2	1 part in 1 $\frac{1}{2}$
Calcium hydroxide (slackened lime).....	$\text{Ca}(\text{OH})_2$	1 " 600
Calcium oxide (unslackened lime).....	CaO	1 " 800
Calcium sulphate.....	CaSO_4	1 " 500
Calcium sulphite.....	CaSO_3	Insoluble
Copper sulphate.....	CuSO_4	1 part in 4
Iron oxide, hydroxide, and sulphide.....		Insoluble
Iron sulphate (coppersas).....	FeSO_4	1 part in 4
Lead acetate.....	$\text{Pb}(\text{C}_2\text{H}_3\text{O}_2)_2$	1 " 2
Lead carbonate.....	PbCO_3	Insoluble
Lead oxide (litharge).....	PbO	"
Lead sulphate.....	PbSO_4	"
Lead sulphite.....	PbSO_3	"
Lead chloride.....	PbCl_2	1 part in 93
Magnesium sulphate.....	MgSO_4	1 " 3
Mercuric chloride.....	HgCl_2	1 " 15
Oxalic acid.....	$\text{C}_2\text{O}_4\text{H}_2\cdot 2\text{H}_2\text{O}$	1 " 10 $\frac{1}{2}$
Potassium bicarbonate.....	KHCO_3	1 " 3
Potassium carbonate.....	K_2CO_3	1 " 1
Potassium cyanide.....	KCN	1 " $\frac{1}{2}$
Potassium ferrocyanide.....	$\text{K}_4\text{Fe}(\text{CN})_6$	1 " 3 $\frac{1}{2}$
Potassium hydroxide.....	KOH	1 " 1
Potassium iodide.....	KI	1 " $\frac{1}{2}$
Potassium sulphate.....	K_2SO_4	1 " 9
Silver nitrate.....	AgNO_3	1 " $\frac{1}{2}$
Sodium bicarbonate.....	NaHCO_3	1 " 10
Sodium bisulphate.....	NaHSO_4	1 " 3 $\frac{1}{2}$
Sodium carbonate.....	Na_2CO_3	1 " 4
Sodium hydroxide.....	NaOH	1 " 1
Sodium sulphate.....	Na_2SO_4	1 " 4
Zinc carbonate.....	ZnCO_3	Insoluble
Zinc cyanide.....	$\text{Zn}(\text{CN})_2$	"
Zinc hydroxide.....	$\text{Zn}(\text{OH})_2$	"
Zinc sulphate.....	ZnSO_4	1 part in 1 $\frac{1}{2}$

¹ Boiling water.

CAPACITIES OF TANKS.

Diam- eter.		Per foot depth.					Diam- eter.	
Ft.	In.	Cubic feet.	Gallons.	Tons water.	Tons sand. ¹	Tons water per inch depth.	Ft.	In.
4	0	12.57	93.97	0.39	0.50	0.033	4	0
4	6	15.90	118.93	0.50	0.64	0.041	4	6
5	0	19.64	146.83	0.61	0.79	0.051	5	0
5	6	23.76	177.67	0.74	0.95	0.062	5	6
6	0	28.27	211.44	0.88	1.13	0.074	6	0
6	6	33.18	248.15	1.03	1.33	0.086	6	6
7	0	38.48	287.80	1.20	1.54	0.100	7	0
7	6	44.18	330.38	1.38	1.77	0.115	7	6
8	0	50.27	375.90	1.57	2.01	0.131	8	0
8	6	56.75	424.36	1.77	2.27	0.149	8	6
9	0	63.62	475.75	1.98	2.55	0.165	9	0
9	6	70.88	530.08	2.22	2.84	0.185	9	6
10	0	78.54	587.35	2.46	3.14	0.206	10	0
10	6	86.59	647.55	2.71	3.46	0.225	10	6
11	0	95.03	710.69	2.97	3.80	0.248	11	0
11	6	103.87	776.77	3.25	4.16	0.271	11	6
12	0	113.10	845.18	3.54	4.52	0.295	12	0
13	0	132.73	992.62	4.14	5.31	0.345	13	0
14	0	153.94	1,151.21	4.81	6.16	0.401	14	0
15	0	176.72	1,321.54	5.52	7.07	0.460	15	0
16	0	201.06	1,503.62	6.28	8.04	0.524	16	0
17	0	226.98	1,697.45	7.09	9.08	0.591	17	0
18	0	254.47	1,903.02	7.92	10.18	0.660	18	0
19	0	283.53	2,120.34	8.86	11.34	0.738	19	0
20	0	314.16	2,349.41	9.82	12.57	0.818	20	0
21	0	346.30	2,590.22	10.82	13.85	0.902	21	0
22	0	380.13	2,842.79	11.88	15.21	0.990	22	0
23	0	415.48	3,107.10	12.98	16.62	1.082	23	0
24	0	452.39	3,383.15	14.14	18.10	1.178	24	0
25	0	490.88	3,670.95	15.34	19.64	1.278	25	0
26	0	530.93	3,970.50	16.56	21.24	1.380	26	0
27	0	572.56	4,281.80	17.89	22.90	1.491	27	0
28	0	615.75	4,604.85	19.24	24.63	1.604	28	0
29	0	660.52	4,939.64	20.64	26.42	1.720	29	0
30	0	706.86	5,286.18	22.09	28.27	1.841	30	0
32	0	804.25	6,014.48	25.13	32.17	2.094	32	0
34	0	907.92	6,789.80	28.37	36.32	2.364	34	0
36	0	1,017.88	7,612.08	31.69	40.72	2.641	36	0
38	0	1,134.12	8,481.36	35.44	45.36	2.953	38	0
40	0	1,256.64	9,397.64	39.27	50.27	3.273	40	0
42	0	1,385.45	10,360.88	43.30	55.42	3.608	42	0
44	0	1,520.53	11,371.16	47.52	60.82	3.960	44	0
46	0	1,661.91	12,428.40	51.94	66.48	4.328	46	0
48	0	1,809.56	13,532.60	56.55	72.38	4.712	48	0
50	0	1,963.50	14,683.80	61.36	78.54	5.113	50	0

¹ Figures for tons of sand are based on 25 cubic feet per ton.

SLIME PULP FORMULA (*Caldecott*).

Result sought.	Per cent by weight of dry slime in pulp.	Per cent by weight of solution in pulp.	Ratio by weight of solution to one of dry slime.	Volume in cu. ft. of 1 ton of pulp.	Weight in lb. of dry slime in 1 cu. ft. of pulp.	Weight in lb. of dry slime in 1 cu. ft. of pulp.	Volume in cu. ft. of pulp containing 1 ton of slime.	Weight in tons of dry slime in charge of pulp.
When specific gravity of dry slime = S .	$\frac{100 S (z - 1)}{z (S - 1)}$	$\frac{100 (S - z)}{z (S - 1)}$	$\frac{S - z}{S (z - 1)}$	$\frac{32}{z}$	$\frac{1000 z}{16}$	$\frac{62.5 z (z - 1)}{S - 1}$	$\frac{32 (S - 1)}{S (z - 1)}$	$Cu. ft. pulp \times \frac{S (z - 1)}{32 (S - 1)}$
When specific gravity of dry slime = 2.7	$\frac{158.824 (z - 1)}{z}$	$\frac{100 - 58.824 (z - 1)}{z}$	$\frac{1 - 37 z}{z - 1}$	$\frac{32}{z}$	$62.5 z$	$90.245 (z - 1)$	$\frac{20.148}{z - 1}$	$Cu. ft. pulp \times \frac{z - 1}{20.148}$

1. Let S. G. of slime pulp = z .
2. Let per cent (by weight) of dry slime in pulp = $\frac{S - z}{S}$.
3. One cubic foot of water or solution = 62.5 lb.
4. One ton of water or solution = 32 cu. ft.
5. Capacity in cubic feet of circular vat = (diam. in ft.)² \times (depth in feet)² $\times \frac{\pi}{4}$.
6. Specific gravity of alum pulp = $\frac{\text{Weight of (flask + pulp)}}{\text{Weight of (flask + water)}} - \text{weight of flask}$.
7. Specific gravity of dry slime = $\frac{\text{Weight of dry slime in air}}{\text{Weight of dry slime in air} + \text{weight of flask + water}} = \frac{\text{Weight of dry slime in air}}{\frac{z^3}{27} - 100 (z - 1)}$.
8. One gram per liter = 1 os. (avord.) per cubic foot.

Notes. — In sampling a charge of pulp undergoing agitation, a narrow necked vessel fastened to a rod should be employed, so that the sample may be taken not only from the surface of the charge but also from as far below it as possible. The 500-c.c. flask is a convenient size to employ in determining the specific gravity of the pulp, and the vessel should be well shaken in the intervals of pouring into the flask, so as to prevent settlement of the slime.

SLIME DENSITY RELATIONS.

Specific gravity of pulp and volume of one ton in cubic feet, for slimes containing solids of different specific gravities.

Per cent solids.	Ratio of solids to solution.	2.50		2.60		2.70		2.80		2.90	
		S.G.	Vol.								
5	1:19.000	1031	31.03	1032	31.01	1032	31.01	1033	30.97	1034	30.96
6	1:15.667	1037	30.85	1036	30.82	1039	30.79	1040	30.76	1041	30.74
7	1:13.286	1044	30.66	1045	30.62	1046	30.59	1047	30.56	1048	30.53
8	1:11.500	1050	30.46	1052	30.43	1053	30.39	1055	30.36	1055	30.32
9	1:10.111	1057	30.27	1059	30.23	1060	30.19	1061	30.16	1063	30.11
10	1:9.000	1064	30.08	1065	30.03	1067	29.99	1068	29.95	1070	29.90
11	1:8.091	1071	29.88	1073	29.83	1074	29.79	1076	29.74	1078	29.69
12	1:7.233	1078	29.70	1080	29.64	1082	29.59	1083	29.53	1085	29.49
13	1:6.692	1085	29.50	1087	29.44	1089	29.39	1091	29.33	1093	29.27
14	1:6.144	1092	29.31	1094	29.24	1097	29.19	1099	29.12	1101	29.06
15	1:5.667	1099	29.18	1102	29.06	1104	28.99	1107	28.91	1109	28.85
16	1:5.250	1106	28.93	1109	28.85	1112	28.78	1115	28.71	1117	28.65
17	1:4.882	1114	28.74	1117	28.65	1119	28.58	1123	28.50	1125	28.44
18	1:4.556	1121	28.54	1125	28.45	1128	28.38	1131	28.30	1134	28.23
19	1:4.263	1129	28.35	1133	28.26	1136	28.18	1139	28.09	1142	28.03
20	1:4.000	1136	28.17	1140	28.06	1144	27.98	1147	27.89	1151	27.81
21	1:3.762	1144	27.97	1148	27.87	1152	27.77	1156	27.68	1159	27.60
22	1:3.545	1152	27.78	1157	27.67	1161	27.57	1165	27.47	1168	27.39
23	1:3.348	1160	27.58	1165	27.47	1169	27.37	1174	27.27	1177	27.18
24	1:3.167	1168	27.39	1173	27.27	1178	27.17	1182	27.06	1186	26.97
25	1:3.000	1176	27.21	1182	27.06	1187	26.97	1191	26.85	1195	26.76
26	1:2.846	1185	27.01	1190	26.89	1195	26.77	1201	26.65	1206	26.55

SLIME DENSITY RELATIONS (*Continued*).

Specific gravity of pulp and volume of one ton in cubic feet, for slimes containing solids of different specific gravities.

Per cent solids.	Ratio of solids to solution.	2.50		2.60		2.70		2.80		2.90	
		S.G.	Vol.								
27	1:2.704	1193	26.82	1199	26.68	1205	26.56	1210	26.44	1215	26.34
28	1:2.571	1203	26.62	1209	26.49	1214	26.36	1220	26.24	1224	26.13
29	1:2.448	1211	26.43	1217	26.29	1223	26.16	1229	26.03	1234	26.92
30	1:2.333	1220	26.24	1226	26.10	1233	25.95	1239	25.83	1244	25.71
31	1:2.226	1229	26.06	1236	26.90	1242	25.75	1249	25.63	1255	25.50
32	1:2.125	1238	25.86	1245	25.70	1252	25.55	1259	25.42	1265	25.29
33	1:2.030	1247	25.66	1256	25.50	1262	25.35	1269	25.21	1276	25.16
34	1:1.940	1256	25.47	1264	25.31	1272	25.15	1279	25.01	1287	24.87
35	1:1.857	1266	25.28	1274	25.12	1283	24.95	1290	24.80	1298	24.66
36	1:1.778	1276	25.09	1284	24.91	1293	24.75	1301	24.60	1309	24.45
37	1:1.703	1285	24.90	1296	24.71	1304	24.55	1312	24.39	1320	24.24
38	1:1.632	1295	24.70	1306	24.52	1314	24.36	1323	24.19	1332	24.03
39	1:1.564	1305	24.51	1316	24.32	1326	24.14	1335	23.98	1343	23.82
40	1:1.500	1316	24.32	1326	24.13	1336	23.95	1346	23.77	1355	23.61
41	1:1.439	1326	24.13	1337	23.93	1348	23.74	1357	23.57	1367	23.40
42	1:1.381	1337	23.94	1348	23.73	1359	23.55	1370	23.36	1380	23.19
43	1:1.326	1348	23.74	1359	23.53	1371	23.34	1382	23.16	1393	22.99
44	1:1.273	1359	23.55	1372	23.33	1383	23.15	1396	22.95	1406	22.78
45	1:1.222	1370	23.36	1383	23.14	1395	22.94	1407	22.74	1418	22.57
46	1:1.174	1381	23.17	1396	23.04	1408	22.73	1420	22.54	1432	22.36
47	1:1.128	1393	22.98	1407	22.75	1420	22.54	1433	22.33	1445	22.15
48	1:1.083	1404	22.78	1419	22.53	1433	22.33	1446	22.12	1453	21.98

USEFUL DATA

SLIME DENSITY RELATIONS (Continued).

Specific gravity of pulp and volume of one ton in cubic feet, for slimes containing solids of different specific gravities.

Per cent solids.	Ratio of solids to solution.	2.50			2.60			2.70			2.80			2.90			
		S.G.	Vol.	S.G.	Vol.	S.G.	Vol.	S.G.	Vol.	S.G.	S.G.	Vol.	S.G.	S.G.	Vol.		
49	1:1.041	1416	22.59	1431	22.35	1446	22.13	1460	21.92	1473	21.73	1487	21.63	1502	21.51	1517	21.40
50	1:1.000	1429	22.39	1444	22.16	1460	21.92	1473	21.71	1488	21.51	1502	21.31	1517	20.89	1532	20.89
51	1:1.0961	1441	22.21	1458	21.96	1473	21.73	1487	21.52	1502	21.30	1516	21.10	1532	20.89	1548	20.68
52	1:1.0923	1453	22.02	1471	21.76	1487	21.52	1501	21.32	1516	21.10	1532	20.89	1548	20.47	1564	20.47
53	1:1.0887	1466	21.82	1484	21.56	1501	21.32	1516	21.10	1532	20.89	1548	20.26	1564	20.26	1580	20.05
54	1:1.0852	1479	21.63	1498	21.36	1515	21.12	1530	20.92	1547	20.69	1563	20.48	1580	20.27	1596	20.05
55	1:1.0809	1493	21.44	1512	21.17	1530	20.92	1547	20.69	1563	20.48	1580	20.27	1596	20.05	1613	19.84
56	1:1.0786	1506	21.25	1526	20.97	1545	20.72	1560	20.51	1579	20.27	1596	20.05	1620	19.86	1639	19.63
57	1:1.0754	1520	21.06	1540	20.77	1560	20.51	1574	20.31	1595	20.07	1613	19.84	1639	19.63	1658	19.42
58	1:1.0724	1534	20.86	1555	20.68	1574	20.31	1595	20.11	1611	19.86	1628	19.66	1645	19.45	1664	19.21
59	1:1.0695	1548	20.67	1572	20.38	1591	20.18	1607	19.91	1623	19.66	1645	19.45	1663	19.26	1683	18.79
60	1:1.0667	1563	20.48	1585	20.18	1611	19.98	1623	19.71	1641	19.51	1662	19.30	1681	19.04	1703	18.79
61	1:1.0639	1577	20.29	1601	19.98	1635	19.71	1651	19.51	1671	19.30	1691	18.94	1713	18.58	1732	18.37
62	1:1.0613	1592	20.10	1617	19.79	1641	19.51	1667	19.30	1681	19.04	1698	18.84	1718	18.73	1738	18.53
63	1:1.0587	1606	19.90	1633	19.59	1657	19.30	1675	19.10	1692	18.90	1718	18.73	1738	18.53	1757	18.32
64	1:1.0563	1623	19.71	1650	19.40	1675	19.10	1698	18.94	1723	18.73	1757	18.53	1783	18.32	1803	18.11
65	1:1.0538	1639	19.52	1667	19.20	1692	18.90	1718	18.70	1744	18.50	1776	18.32	1803	18.11	1825	17.89
66	1:1.0515	1656	19.32	1684	19.00	1711	18.70	1738	18.50	1767	18.32	1803	18.11	1825	17.89	1847	17.52
67	1:1.0493	1672	19.14	1701	18.80	1730	18.50	1757	18.30	1786	18.10	1813	17.81	1847	17.52	1870	17.32
68	1:1.0471	1689	18.94	1719	18.61	1749	18.30	1776	18.10	1803	17.81	1830	17.52	1858	17.32	1887	17.11
69	1:1.0449	1706	18.75	1738	18.41	1766	18.21	1794	17.90	1830	17.60	1858	17.32	1887	17.11	1917	16.90
70	1:0.429	1724	18.56	1767	18.21	1796	17.90	1813	17.60	1847	17.32	1887	17.11	1917	16.90	1947	16.70

Per cent solids.	Ratio of solids to solutions.	3.00			3.10			3.20			3.30			3.40		
		S.G.	Vol.													
5	1:19.000	1035	30.93	1035	30.92	1036	30.90	1036	30.89	1040	30.76	1040	30.51	1049	30.51	
6	1:15.667	1042	30.72	1042	30.70	1043	30.68	1043	30.66	1049	30.51	1049	30.51	1049	30.51	
7	1:13.286	1049	30.51	1049	30.48	1050	30.46	1050	30.43	1055	30.26	1055	30.26	1058	30.26	
8	1:11.500	1056	30.30	1057	30.27	1058	30.24	1058	30.21	1059	30.01	1067	29.76	1067	29.76	
9	1:10.111	1064	30.09	1065	30.05	1066	30.02	1066	29.99	1075	29.51	1084	29.51	1084	29.51	
10	1:9.000	1071	29.87	1072	29.83	1074	29.80	1074	29.77	1075	29.36	1063	29.36	1063	29.36	
11	1:8.091	1078	29.65	1080	29.61	1082	29.58	1082	29.54	1083	29.01	1102	29.01	1102	29.01	
12	1:7.333	1087	29.44	1088	29.40	1090	29.36	1090	29.32	1091	28.10	1112	28.10	1112	28.10	
13	1:6.692	1095	29.23	1096	29.18	1098	29.14	1098	29.10	1109	28.52	1122	28.52	1122	28.52	
14	1:6.144	1103	29.01	1105	28.96	1106	28.92	1106	28.88	1108	28.27	1132	28.27	1132	28.27	
15	1:5.667	1111	28.80	1113	28.74	1115	28.70	1115	28.66	1117	28.02	1142	28.02	1142	28.02	
16	1:5.250	1119	28.59	1122	28.53	1124	28.48	1124	28.43	1125	27.77	1152	27.77	1152	27.77	
17	1:4.882	1128	28.37	1130	28.31	1132	28.26	1132	28.21	1134	27.52	1163	27.52	1163	27.52	
18	1:4.556	1136	28.16	1139	28.10	1141	28.04	1143	27.99	1143	27.27	1173	27.27	1173	27.27	
19	1:4.203	1145	27.95	1148	27.88	1150	27.82	1153	27.76	1162	27.02	1184	27.02	1184	27.02	
20	1:4.000	1154	27.73	1157	27.66	1159	27.60	1160	27.54	1171	26.77	1194	26.77	1194	26.77	
21	1:3.762	1163	27.52	1166	27.44	1169	27.38	1173	27.32	1181	26.53	1206	26.53	1206	26.53	
22	1:3.545	1172	27.31	1175	27.23	1178	27.16	1181	27.09	1191	26.28	1218	26.28	1218	26.28	
23	1:3.348	1181	27.09	1184	27.01	1188	26.94	1191	26.87	1201	26.03	1230	26.03	1230	26.03	
24	1:3.167	1190	26.88	1194	26.79	1198	26.72	1204	26.50	1211	25.42	1241	25.42	1241	25.42	
25	1:3.000	1200	26.67	1204	26.58	1208	26.50	1214	26.37	1222	26.20	1253	26.20	1253	26.20	
26	1:2.846	1210	26.45	1214	26.37	1218	26.28									

¹ 80 per cent pyrite and 20 per cent quartz.

SLIME DENSITY RELATIONS (Continued).

Specific gravity of pulp and volume of one ton in cubic feet, for slimes containing solids of different specific gravities.

Per cent solids.	Ratio of solids to solutions.	3.00						3.10						3.20						3.30						3.40													
		S.G.	Vol.																																				
27	1:2.704	1220	26.24	1224	26.15	1228	26.06	1232	25.98	1236	25.75	1240	25.53	1244	25.31	1248	25.08	1252	24.86	1256	24.64	1260	24.45	1264	24.24	1268	24.04	1272	23.83										
28	1:2.671	1230	26.03	1234	25.93	1238	25.84	1242	25.62	1246	25.40	1250	25.18	1254	24.96	1258	24.74	1262	24.52	1266	24.31	1270	24.09	1274	23.88	1278	23.67	1282	23.46	1286	23.25								
29	1:2.448	1240	26.81	1244	25.71	1248	25.60	1252	25.49	1256	25.38	1260	25.16	1264	24.95	1268	24.73	1272	24.52	1276	24.31	1280	24.09	1284	23.88	1288	23.67	1292	23.46	1296	23.25								
30	1:2.333	1250	25.60	1255	25.50	1258	25.40	1262	25.28	1266	25.18	1270	24.96	1274	24.75	1278	24.54	1282	24.33	1286	24.12	1290	23.91	1294	23.69	1298	23.48	1302	23.27	1306	23.06								
31	1:2.226	1261	25.39	1266	25.28	1270	25.06	1274	24.96	1278	24.86	1282	24.75	1286	24.65	1290	24.54	1294	24.43	1298	24.32	1302	24.21	1306	24.10	1310	23.99	1314	23.88	1318	23.77	1322	23.66						
32	1:2.125	1271	25.17	1277	25.06	1281	24.96	1285	24.86	1289	24.75	1293	24.64	1297	24.53	1301	24.42	1305	24.31	1309	24.20	1313	24.09	1317	23.98	1321	23.87	1325	23.76	1329	23.65	1333	23.54						
33	1:2.030	1282	24.96	1288	24.86	1292	24.75	1296	24.63	1300	24.52	1304	24.41	1308	24.30	1312	24.19	1316	24.08	1320	23.97	1324	23.86	1328	23.75	1332	23.64	1336	23.53	1340	23.42	1344	23.31						
34	1:1.940	1293	24.75	1299	24.63	1303	24.52	1307	24.41	1311	24.30	1315	24.19	1319	24.08	1323	23.97	1327	23.86	1331	23.75	1335	23.64	1339	23.53	1343	23.42	1347	23.31	1351	23.20	1355	23.09						
35	1:1.857	1304	24.53	1310	24.41	1314	24.30	1318	24.19	1322	24.08	1326	23.97	1330	23.86	1334	23.75	1338	23.64	1342	23.53	1346	23.42	1350	23.31	1354	23.20	1358	23.09	1362	22.98	1366	22.87						
36	1:1.778	1316	24.32	1322	24.19	1326	24.08	1330	23.97	1334	23.86	1338	23.75	1342	23.64	1346	23.53	1350	23.42	1354	23.31	1358	23.20	1362	23.09	1366	22.98	1370	22.87	1374	22.76	1378	22.65						
37	1:1.703	1328	24.11	1334	23.98	1338	23.86	1342	23.76	1346	23.64	1350	23.53	1354	23.42	1358	23.31	1362	23.20	1366	23.09	1370	22.98	1374	22.87	1378	22.76	1382	22.65	1386	22.54	1390	22.43						
38	1:1.632	1340	23.89	1346	23.76	1350	23.64	1354	23.53	1358	23.42	1362	23.31	1366	23.20	1370	23.09	1374	22.98	1378	22.87	1382	22.76	1386	22.65	1390	22.54	1394	22.43	1398	22.32	1402	22.21						
39	1:1.564	1351	23.68	1358	23.56	1362	23.45	1366	23.33	1370	23.22	1374	23.11	1378	23.00	1382	22.89	1386	22.78	1390	22.67	1394	22.56	1398	22.45	1402	22.34	1406	22.23	1410	22.12	1414	22.01	1418	21.90				
40	1:1.600	1363	23.47	1371	23.35	1376	23.26	1380	23.15	1384	23.04	1388	22.93	1392	22.82	1396	22.71	1400	22.60	1404	22.49	1408	22.38	1412	22.27	1416	22.16	1420	22.05	1424	21.94	1428	21.83						
41	1:1.439	1376	23.26	1384	23.15	1389	23.04	1393	22.93	1397	22.82	1401	22.71	1405	22.60	1409	22.49	1413	22.38	1417	22.27	1421	22.16	1425	22.05	1429	21.94	1433	21.83	1437	21.72	1441	21.61	1445	21.50				
42	1:1.381	1389	23.04	1397	22.93	1402	22.82	1406	22.71	1410	22.60	1414	22.49	1418	22.38	1422	22.27	1426	22.16	1430	22.05	1434	21.94	1438	21.83	1442	21.72	1446	21.61	1450	21.50	1454	21.39	1458	21.28				
43	1:1.326	1402	22.83	1410	22.71	1414	22.60	1418	22.49	1422	22.38	1426	22.27	1430	22.16	1434	22.05	1438	21.94	1442	21.83	1446	21.72	1450	21.61	1454	21.50	1458	21.39	1462	21.28	1466	21.17	1470	21.06	1474	20.95		
44	1:1.273	1416	22.61	1424	22.49	1430	22.38	1436	22.27	1442	22.16	1448	22.05	1454	21.94	1460	21.83	1466	21.72	1472	21.61	1478	21.50	1484	21.39	1490	21.28	1496	21.17	1502	21.06	1508	20.95	1514	20.84				
45	1:1.223	1429	22.40	1437	22.19	1443	22.19	1449	22.07	1455	21.97	1461	21.86	1467	21.75	1473	21.64	1479	21.53	1485	21.42	1491	21.31	1497	21.20	1503	21.09	1509	20.98	1515	20.87	1521	20.76	1527	20.65	1533	20.54		
46	1:1.174	1443	22.19	1449	22.19	1455	21.97	1461	21.86	1467	21.75	1473	21.64	1479	21.53	1485	21.42	1491	21.31	1497	21.20	1503	21.09	1509	20.98	1515	20.87	1521	20.76	1527	20.65	1533	20.54	1539	20.43				
47	1:1.128	1457	21.97	1463	21.86	1469	21.75	1475	21.64	1481	21.53	1487	21.42	1493	21.31	1499	21.20	1505	21.09	1511	20.98	1517	20.87	1523	20.76	1529	20.65	1535	20.54	1541	20.43	1547	20.32	1553	20.21	1559	20.10		
48	1:1.063	1471	21.76	1477	21.76	1483	21.65	1489	21.54	1495	21.43	1501	21.32	1507	21.21	1513	21.10	1519	20.99	1525	20.88	1531	20.77	1537	20.66	1543	20.55	1549	20.44	1555	20.33	1561	20.22	1567	20.11	1573	20.00	1579	19.89

1:80 per cent pyrite and 20 per cent quartz.

SLIME DENSITY RELATIONS (Continued).

Per cent solids.	Ratio of solids to solution.	Specific gravity of pulp and volume of one ton in cubic feet, for slimes containing solids of different specific gravities.						14.50	
		3.00	3.10	S.G.	Vol.	S.G.	Vol.	S.G.	Vol.
49	1:1.041	1485	21.55	1497	21.38	1508	21.22	1519	21.07
50	1:1.000	1500	21.33	1512	21.16	1524	21.00	1535	20.85
51	1:0.961	1515	21.12	1528	20.94	1540	20.78	1551	20.62
52	1:0.923	1531	20.91	1544	20.73	1556	20.56	1568	20.40
53	1:0.887	1547	20.69	1560	20.51	1573	20.34	1585	20.18
54	1:0.852	1563	20.48	1577	20.29	1590	20.12	1603	19.96
55	1:0.809	1579	20.27	1594	20.08	1608	19.90	1621	19.73
56	1:0.786	1596	20.05	1611	19.87	1626	19.68	1640	19.51
57	1:0.754	1613	19.84	1628	19.65	1645	19.46	1659	19.29
58	1:0.724	1631	19.63	1646	19.43	1663	19.24	1678	19.06
59	1:0.695	1649	19.41	1665	19.21	1682	19.02	1697	18.84
60	1:0.667	1667	19.20	1684	19.00	1702	18.80	1718	18.62
61	1:0.639	1686	18.98	1704	18.78	1722	18.58	1739	18.39
62	1:0.613	1705	18.77	1724	18.56	1742	18.36	1761	18.17
63	1:0.587	1724	18.56	1745	18.34	1764	18.14	1783	17.95
64	1:0.563	1745	18.35	1765	18.12	1786	17.92	1805	17.72
65	1:0.538	1765	18.13	1786	17.91	1808	17.70	1828	17.50
66	1:0.515	1786	17.92	1808	17.69	1830	17.48	1852	17.28
67	1:0.493	1808	17.71	1831	17.47	1853	17.26	1876	17.06
68	1:0.471	1830	17.49	1854	17.26	1877	17.04	1901	16.83
69	1:0.449	1852	17.28	1878	17.04	1902	16.82	1927	16.61
70	1:0.429	1875	17.07	1902	16.83	1926	16.60	1953	16.39

¹ 80 per cent pyrite and 20 per cent quartz.

MILL NOTES AND DATA.

(From General Engineering Co.'s Ore Testing Bulletin.)

It is to be understood that the following figures are only approximate, and are intended for use in making preliminary estimates only. Exact figures can only be given when all the conditions of a case are known.

POWER.

Crushers — *Blake* — 7×10, 8 H. P.; 9×15, 15 H. P.; 10×20, 20 H. P.; 15×24, 30 H. P.

Dodge — 4×6, 2 H. P.; 7×10, 7 H. P.; 11×15, 15 H. P.

Gates — D Style No. 1, 10 H. P.; No. 2, 15 H. P.; No. 3, 25 H. P.; No. 4, 30 H. P.; No. 5, 40 H. P.; No. 6, 60 H. P.; No. 7, 125 H. P.; No. 8, 150 H. P.

Mills — *Gravity Stamp* — 10-stamp, 90 8-in. drops per minute, 750-lb., 15 H. P.; 850-lb., 17 H. P.; 950-lb., 19 H. P.; 1000-lb., 20 H. P.

Tubes — 5×14-ft., 30 H. P.; 5×22-ft., 70 H. P.; 4×20-ft., 50 H. P.

Chilian — 4-ft., 6 H. P.; 5-ft., 12 H. P.; 6-ft., 25 H. P.

Huntington — 3½-ft., 4-5 H. P.; 5-ft., 6-7 H. P.; 6-ft., 8-10 H. P.

Rolls — 12×20, 12 H. P.; 14×27, 16 H. P.; 16×36, 25 H. P.

Sample Grinders — No. 1, 3 H. P.; No. 2, 4 H. P.

Amalgamating Pans — 5-ft., 4 H. P.; 8-ft., 6 H. P.

Grinding Pans — 5-ft., 6 H. P.; 8-ft., 9 H. P.

Settlers — 8-ft., 3 H. P. **Agitators** — 8-ft., 3 H. P.

Clean-up Pans — 48-in., 1½ H. P.

Hendryx Agitator — 5 H. P.

Revolving Screens — ½ H. P. **Revolving Dryers** — 1 H. P.

Wilfley Tables — ¼ H. P. **Frue Vanners** — ½ H. P.

Hancock Jigs — 5 H. P. **Harz Jigs** — (per compt.) ½ H. P.

Stamps and Pulverizers — For each 800-1000-lb. stamp from 3 to 6 gal. per min. per stamp. On medium hard ores with 1000-lb. stamps crushing to 20 mesh, 5 tons per stamp with 3½ to 4 gal. per min. is good work. This equals a pulp of 4 or 5 to 1. *Chilian* and

Huntington mills on similar ore will require a total of 5 to 6 tons of water per ton of ore passing the screens.

Jigs — For each 18×36-in. compartment of Harz Jigs treating:

- 1½—2½ m.m. particles, 2—4 gal. per min.
- 3½—5 m.m. particles, 5—7 gal. per min.
- 7—10 m.m. particles, 8—14 gal. per min.
- 15—20 m.m. particles, 21—28 gal. per min.

This is in addition to the water in the feed, which can be assumed to be not less than 1—1 water and ore.

Revolving Screens (spray water) — Each 36×72 screen will require on:

- 1½—3½ m.m. holes, 15—25 gal. per min.
- 3½—7 m.m. holes, 10—15 gal. per min.
- 7—15 m.m. holes, 5—8 gal. per min.

Callow T. B. Screens — (24-in. Duplex) —

Water in pulp = 3½—4 tons water per ton of feed.

Shaking spray, 6—10 gal. per min.

Oversize spray, 8—12 gal. per min.

Wilfley Tables — 5—10 gal. per min.

Six-foot Frue Vanners — 1½—3 gal. per min.

Hydraulic Classifiers — A very variable quantity.

Richards gives from 10—50 tons water per ton of pulp, or an average of 20 tons for all the mills visited by him.

Wilfley Tables — Tons per 24 hr.:

On unclassified feeds, 20-mesh and finer, 10—15.

On screen-sized feeds, 30 to 120-mesh, inclusive, 25—35.

On minus 120-mesh feeds, 15—20.

On minus 200-mesh slimes, 4—6.

Frue Vanners, 6-ft. — Tons per 24 hr.:

On stamp-mill pulps, 20-mesh and finer, 6—10.

On minus 200-mesh slimes, 3—5.

Buddles, 16-ft. — Tons per 24 hr. per deck, 6—10.

Callow Settling Tanks — Treating dilute slime overflows

(1 per cent solids) and giving a clear overflow and a discharge of 15—25 per cent solids:

On Butte slimes, 25—30 gal. per min. of feed.

On Cœur d'Alene slimes, 30—35 gal. per min. of feed.

On Bingham porphyry slimes, 35—40 gal. per min. of feed.

As Wilfley feed tanks on minus 30-mesh materials 50 tons of solids per day contained in 35-40 gal. per min. (= 4½-5 to 1) settling will equal 33 per cent solids and overflow 3.5 per cent solids.

COST PER TON OF TREATMENT BY DIFFERENT MILLS.

Stamps and Vanners —

California.....	\$0.40-\$0.50 in	10-	40 stamp units.
Black Hills.....	0.40- 0.50 in	200	stamp units.
Alaska.....	0.28- 0.30 in	200	stamp units.
Gilpin County.....	0.75- 1.00 in	10-	40 stamp units.

Coarse Concentration, Jigs, Rolls and Tables —

Missouri.....	\$0.25-\$0.30 in	100-	400 ton units.
Colorado and Utah..	0.75- 1.00 in	100-	150 ton units.
Large Montana plants	0.35- 0.50 in	1000-2000	ton units.
Lake Superior steam- plants.....	0.25- 0.35 in	1000-2000	ton units.

Combination Mills —

Wet stamping, van- ners and pans.....	\$1.50-\$5.00 in	60-100	stamp units.
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Chloridizing Mills —

Dry stamping, roast- ing and pans.....	\$6.00-\$7.00 in	60-100	stamp units.
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Magnetic Separators —

Magnet iron ocs.....	\$0.18-\$0.50 in	300-400	ton units.
Wetherill plants.....	0.40- 0.75 in	200-300	ton units.
Wilfley roasters.....	0.75- 1.00 in	50-100	ton units.

Cyanide Mills —

Dry crushing with rolls and leaching	\$1.50-\$2.50 in	100-200	ton units.
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Wet crushing, leach- ing and filter press- ing, or wet crush- ing, sliming and filter pressing.....	\$0.85-\$2.00 in	100-200	ton units.
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Chlorination Mills —

Barrel process.....	\$3.50-\$5.00 in	150-200	ton units.
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WATER REQUIRED PER TON OF ORE 239

HORSE-POWER PER TON OF ORE TREATED PER DAY BY DIFFERENT MILLS.

Stamps and Vanners — 20-40 mesh, 0.75-1.0.
Coarse Concentration — 10-20 mesh, 0.5-0.8.
Combination Stamp Mills — 16-30 mesh, 1.5-1.75.
Chloridizing Stamp Mills — Dry to 16 mesh, 2-2.5.
Chloridizing Stamp Mills — Wet to 40 mesh, 4-4.5.
Magnetic Separator Mills — 0.25-0.5.
Cyanide Mills — Dry roll crushing to 20-30 mesh, 0.5-0.8.
Cyanide Mills — Wet stamp crushing and sliming to 80 mesh, 0.75-1.5.

COST OF POWER PER H.-P. PER DAY.

Steam power, non-condensing engines, coal at \$4.50, 16-18 cents.

Steam power, condensing engines, coal at \$4.50, 14-16 cents.

Steam power, compound condensing engines, coal at \$4.50, 11-14 cents.

For each \$1 increased cost of coal above \$4.50 per ton, add to the above figures 1½ cents per h.-p. per day for non-condensing, 1½ cents for condensing, and 1 cent for compound condensing engines.

Electric power, \$40-\$65 per h.-p. per annum = 11-18 cents per h.p.-day.

An empirical figure for cost of steam power in mining camps in general is 25 cents per h.-p.-day.

TOTAL QUANTITY OF WATER REQUIRED PER TON OF ORE TREATED IN DIFFERENT MILLS.

Stamps and Vanners — 20-30 mesh, 4-6 tons.
Coarse Concentration — Rolls, jigs and tables, 15-20 tons.
Combination Mills — Stamps, Vanners and Pans, 6-8 tons.
Cyanide Mills — Shoveling Tails and Filter Pressing, 0.2-0.35 tons.
Cyanide Mills — Sluicing Tailings, 1.3-1.9 tons.

These figures are exclusive of water for boilers and allow for no pumping back. With settling tanks or ponds it is safe figuring to say that 50 to 60 per cent of the original water (in all but mills shoveling tailings) is available for use over again, and that with automatically discharging tanks for sands and decantation tanks for slimes, 75 to 80 per cent of the gross quantity used can be recovered.

WATER REQUIRED FOR STEAM ENGINES PER INDICATED HORSE-POWER.

Non-condensing Engines — 0.05–0.066 gal. per min.

Condensing Engines — 0.04–0.05 gal. per min.

Compound Condensing Engines — 0.025–0.04 gal. per minute.

U. S. Gal. per min. \times 500 = Pounds of steam per hour.

One "boiler" horse-power = 30 lb. water evaporated per hour.

= 10 sq. ft. of heating surface.

= $\frac{1}{2}$ sq. ft. of grate surface.

= 13 sq. in. of flue area.

= 6 sq. in. of smoke-stack area.

CAPACITIES.

Crushers — *Blake* — Tons per 24 hr. through 2-in. 7×10 , 120; 9×15 , 190; 10×20 , 300; 15×24 , 420.

Dodge — Tons per 24 hr., 2-in. to $\frac{3}{4}$ -in. 4×6 , 30; 7×10 100; 11×15 , 150.

Gates — Style D. Tons of run of mine per 24 hr. through $2\frac{1}{2}$ -in. No. 1, 95–190; No. 2, 140–280; No. 3, 240–480; No. 4, 360–720; No. 5, 600–960; No. 6, 720–1440; No. 7 $\frac{1}{2}$, 1800–3000; No. 8, 3000–4800.

Stamps — With 1000-lb. California stamps on medium hard ores to 20 mesh, 5 tons per stamp; on hard ores to the same mesh, $3\frac{1}{2}$ tons per stamp is good work. 1200-lb. Nissen stamps on soft Bingham porphyries have crushed 8 tons per stamp through 26 mesh.

Tube-mills — A 4×20 -ft. mill at El Oro grinds 125 tons per day of 20-mesh sands through 100 mesh.

MISCELLANEOUS ITEMS OF MILL COSTS 241

A 4×18 ft. mill at Waihi, N. Z., grinds 77 tons per day of 20-mesh sands through 120 mesh.

A 5×14-ft. mill in Colorado grinds 75 tons per day of 16-mesh sands through 80 mesh.

Rolls — Tons per 24 hr.

Crushing from 1-in. to $\frac{1}{4}$, 12×20, 160; 14×27, 190; 16×36, 225.

Crushing from $\frac{1}{4}$ to 12 mesh, 12×20, 70; 14×27, 80; 16×36, 100.

Crushing from 12 to 24 mesh, 12×20, 35; 14×27, 45; 16×36, 60.

Huntington Mills — Crushing jig tails or mids. from 10. mm. and under, through 16 mesh — tons per 24 hr.: 3 $\frac{1}{2}$ ft., 10-15; 5 ft., 50-75; 6 ft., 75-100.

Chilian Mills of the same nominal dimensions have approximately 25 per cent greater capacity than Huntingtons.

Revolving Screens, 36×72-in. slope 1 $\frac{1}{2}$ -1 $\frac{1}{2}$ in. per ft. 12 r.p.m., screening wet — tons per 24 hr.

Minus 3-m.m. feed on 1 $\frac{1}{2}$ -m.m. round holes, 35- 50.

Minus 6-m.m. feed on 3 -m.m. round holes, 70-100.

Minus 12-m.m. feed on 6 -m.m. round holes, 140-200.

Minus 25-m.m. feed on 12 -m.m. round holes, 280-400.

Callow T. B. Screens (24-in. Duplex) — Tons per 24 hr.: 20 mesh, 250; 30 mesh, 200; 40 mesh, 150; 60 mesh, 125; 80 mesh, 100; 100 mesh, 75; 150 mesh, 50.

MISCELLANEOUS ITEMS OF MILL COSTS.

Stamp Batteries —

Wear and tear 7-12 cents per ton crushed.

Labor 8-16 cents per ton crushed.

Power 8-12 cents per ton crushed.

Wear and tear on dry crushing roll mills, 12-15 cents per ton crushed.

Wear and tear of wet crushing roll mills, 8-10 cents per ton crushed.

Crushing with Huntington mills, 7-25 cents per ton crushed.

Tube-mill grinding, at El Oro, 35 cents; Goldfield, 40 cents; Brodie, 57 cents.

Filter pressing slimes, Moore at Brodie, 30 cents per ton of slimes.

Filter pressing slimes, Butters at Goldfield, 45 cents per ton of slimes.

Filter pressing slimes, Butters at Virginia City, 10 cents per ton of slimes.

Filter pressing slimes, leaf process, W. Australia, 60 cents per ton of slimes.

Screen cloth (4, 6, 18, 30 mesh) in dry roll mills, 2-3 cents per ton.

Screen cloth (10, 20, 30, 50 mesh) in dry roll mills, 4-4½ cents per ton.

Screen cloth (16, 9, 5, 3 m.m.) in wet roll mills (good water) ½-1 cent per ton.

Screen cloth (16, 9, 5, 3 m.m.) in wet roll mills (acid water) ½-1 cent per ton.

Screen cloth, Callow screens, 20-40 mesh, ½-1 cent per ton.

Screen cloth, callow screens. 80-120 mesh, 1-1½ cents per ton.

COST OF ERECTING MILLS, PER TON OF DAILY CAPACITY.

Stamp and vanner mills.....	\$350-400 in 100 ton units.
Coarse concentration mills...	200-300 in 100 ton units.
Cyanide, dry crushing with rolls.....	400-500 in 100 ton units.
Cyanide, wet stamp crushing, sliming and filter-pressing..	400-600 in 100 ton units.

An empirical figure for the erection and installation costs of ore mills and similar reduction plants is \$1 to \$1.50 for each dollar's worth of machinery f.o.b. factory.

Mill buildings, under ordinary mining camp conditions, will cost for each cubic foot of enclosed space:

When of wood, 3-6 cents per cubic foot.

When of steel, 6-10 cents per cubic foot.

Erecting mill and smelter buildings, wood, \$12-15 per M. ft. b. m.

Erecting mill and smelter buildings, steel, 0.5-0.6 cent per pound.

MISCELLANEOUS FORMULAS.

To find diameter of water pipes for mills and smelters.

$$D \text{ (inch)} = \sqrt{\frac{\text{U. S. Gal. per min.} \times 0.41}{\text{Vel. in ft. per sec.}}}$$

Velocity should not exceed 6 as a max.; 4 is mean figure.

Example: 156 gal. per min., vel. say 4 ft. per sec.

$$\sqrt{\frac{156 \times .41}{4}} = \sqrt{16} = 4'' \text{ pipe.}$$

In gradual reduction crushing — to find the total tonnage passing through rolls and to be handled by the elevator returning the roll product back to the sizing screen:

T = Initial tonnage to be crushed per day.

$\% O$ = % of oversize retained on sizing screen after passing through rolls.

Q = Total tonnage to be handled by the elevator and rolls per day.

$$Q = \frac{T}{1 - \% O}$$

Example: $T = 100$ tons, $\% O = 0.75$,

$$Q = \frac{100}{1 - 0.75} = \frac{100}{0.25} = 400 \text{ tons per day.}$$

Recovery and ratio of concentration on an ore; the assay value of the feed, tails and concentrates being known:

H = Feed assay, T = Tailings assay, C = Concentrates assay, R = Ratio of concentration (tons into 1), E = % Recovery.

$$R = \frac{C - T}{H - T} \cdot E = \frac{C \times 100}{H \times R}$$

Example: Feed = 2.4% Pb, Tails = 0.95 % Pb, Concentrates = 11.95% Pb.

$$R = \frac{11.9 - 0.95}{2.4 - 0.95} = \frac{10.95}{1.45} \times 7.56, \text{ Ratio of concentration.}$$

$$E = \frac{11.9 \times 100}{2.4 \times 7.56} = \frac{11.90}{18.15} = 65.8\%, \text{ Recovery.}$$

CARE OF COPPER PLATES.

Never permit your men to walk upon or in any other manner mar the surface of the plates.

Never use acids in dressing your plates; finely pulverized brick dust, wood ashes, sal ammoniac and elbow grease are far better.

Ordinarily plates should be re-dressed every six hours.

A light coating of silver amalgam, well brushed in, makes a very excellent finishing coating.

In running an ore which is acidulated from the decomposition of sulphides, the use, sparingly, of zinc amalgam on the plates, is very beneficial.

Sodium amalgam should never be used when running an ore containing antimony.

Always wash your plates with boiling water after dressing them, as many of the chemicals used for this purpose have deleterious after-effects, such as hardening the plates, etc.

A very good wash for general purposes is as follows: Cyanide of potassium, 2 oz.; caustic soda, 5 oz.; dissolved in 5 gal. of soft water.

When the plates are fouled with black sulphurets, use sal ammoniac, 1 lb.; slaked lime, 3½ lb.; carb. of soda, ½ lb.; water, 1 gal. This should be used as soon as possible after being compounded.

When fouled with zinc, black oxide of manganese, or sulphate of copper, a solution composed of 1 lb. muriatic acid to 5 gal. soft water may be used.

About 2½ oz. of mercury is required to amalgamate and hold 1 oz. of gold in proper condition on the plates.

The ordinary loss of quicksilver to be anticipated is ½ oz. per ton of ore. Greater losses will occur with partly refractory ores.

The grades of the plates may be varied from one inch to three inches to the linear foot, according to circumstances. No rule can be laid down.

ESTIMATION OF INGREDIENTS OF CYANIDE SOLUTIONS

Active Cyanogen Compounds

Free Cyanide. — Liebig's Method: "Free Cyanide" includes such as KCN , $NaCN$, NH_4CN , $CaCN_2$, etc. Universally adopted method for daily tests to regulate strength of solutions used for treatment of ores. In Erlenmeyer flask place known amount of cyanide solution (depending upon strength of solution). Run in standard $AgNO_3$ solution to permanent white turbidity. For dirty, foul mill solutions add KI as indicator, and titrate to yellowish tinge. Standard solutions: Dissolve 13.0464 gr. pure crystallized $AgNO_3$ in distilled water, dilute to 1000 c.c. Each c.c. = 0.01 gr. KCN . With 10 c.c. cyanide solution, each c.c. $AgNO_3$ solution = 0.1% free cyanide. For dilute cyanide solutions, dissolve 6.5232 gr. $AgNO_3$ in liter. Each c.c. this solution = 0.005 gr. KCN . Hence with 50 c.c. cyanide solution, 1 c.c. = 0.01% KCN .

Total Cyanide. — "Total Cyanide" indicates all the cyanogen existing in the form of simple cyanides, hydrocyanic acid, and the double cyanides of zinc, in terms of KCN . Test: To the cyanide solution to be tested add 10 c.c. of 5% $NaOH$. If precipitate is formed filter off 15 c.c. and titrate, after addition of KI , with standard silver solution. If zinc is present, a large excess of alkali should be added; in this case cyanogen found represents potassium cyanide and also the double zinc compound. By estimating the zinc, amount of free KCN may readily be calculated, as 1 part Zn corresponds with 4 parts KCN . If copper is present, a similar allowance must be made. If Ca , Mg , and Mn are present, NH_4Cl must be added, whilst soda is used in presence of Pb and Al . Above titration should be carried to a permanent yellowish color. Reactions — $HCN + NaOH = NaCN + H_2O$; $K_2Zn(CN)_4 + 4 KOH = KCN + Zn(KO)_2 + 2H_2O$.

Total Cyanogen. — "Total Cyanogen" indicates all cyanogen present in the solution, whether it be in the form of simple, double or hydrocyanic acid, ferro- and ferricyanides, sulphocyanides, cyanates and isocyanates. Volumetric Method: two or three drops of saturated solution of neutral (yellow) K_2CrO_4 , are added to solution to be titrated, and standard silver solution run in until a faint reddish tinge appears and remains permanent on shaking. With a pure solution of a simple cyanide, twice as much $AgNO_3$ solution must be added as in the Liebig's method, in order to obtain the reaction, since the indication with chromate does not occur until the whole of the cyanide has been precipitated, after which red silver chromate is precipitated, as: $K_2CrO_4 + 2AgNO_3 = Ag_2CrO_4 + 2KNO_3$. The solution must not contain much free alkali. If present, neutralise with dilute H_2SO_4 and HNO_3 . When HCN is present, neutralise with magnesium carbonate suspended in water. Numerous gravimetric methods are available.

Hydrocyanic Acid. — "Free Hydrocyanic Acid" may be estimated by titration with $AgNO_3$, after the addition of an excess of caustic alkali; after the addition of an excess of bicarbonate. The liquid to be titrated is measured from a burette, dipping into a solution of caustic alkali contained in a beaker. Titrate the alkaline cyanide solution with $AgNO_3$ in the usual way. Avoid large excess of alkali, unless KI indicator is used. If free cyanide is present, titrate first from solution with $AgNO_3$ without caustic alkali. This minus first titration = HCN present in terms of KCN . This method not applicable to solutions containing easily decomposed double cyanides, as $K_2Zn(CN)_4$. Ferro- and ferricyanides do not interfere. Another method applicable in presence of $K_2Zn(CN)_4$: add a solution of K or $NaHCO_3$ (free carbonic acid must be absent), titrate with $AgNO_3$ which gives HCN in terms of KCN .

Available Cyanide. — "Available Cyanide" is that having solvent power on Au and Ag . A method of determining the relative efficiency of solution is by suspending pieces of Au or Ag foil in the solution for equal lengths of time, removing and weighing. The usual method of estimating the available cyanide or dissolving efficiency is to

parative laboratory tests on two portions of the same ore, treating one with mill solution to be tested and the other with a new solution. Care should be taken, that the amount of sample taken, the volume of solution used, and the strength of solution, etc., is exactly similar in the duplicate tests.

Alkaline Constituents

Total Alkali. — "Total Alkali" of a solution is defined as the equivalent, in terms of KOH, of all the ingredients which are alkaline to methyl orange. These are: simple cyanides, hydrates, carbonates, bicarbonates, sulphides, zincates of Na, K, NH₄, Ca, Ba, etc., double cyanides of Zn, free ammonia. Method: a measured volume (50 c.c.) of solution placed in flask, with a few drops of 0.1% solution methyl orange. Titrate with N/10 acid (HCl, HNO₃, or H₂SO₄) to permanent pink tint.

Protective Alkali. — "Protective alkali" is that which will wholly neutralize certain ingredients in an ordinary working solution, as dilute mineral acids, or carbonic acid, before any decomposition of cyanide takes place. Method: To measured volume of solution (10 or 50 c.c.) add AgNO₃ to permanent turbidity. To same solution add a drop of alcoholic 0.5% phenolphthalein without filtering. Titrate the resulting pink fluid with N/10 or any convenient standard acid (oxalic acid is good) until the color entirely disappears. The amount of standard acid used measures the protective alkali. In the presence of zinc, first titrate for total cyanide, and on another portion (50 c.c.) of original solution add an excess of ferrocyanide solution, and then a little more silver solution than was used in the first test, to insure complete conversion of all cyanides into silver salts; add phenolphthalein and titrate with standard acid as above.

Hydrates, Carbonates and Bicarbonates. — Hydrates may be estimated by adding the necessary amount of a standard solution of BaCl₂ to precipitate the carbonate (avoiding an excess which would tend to precipitate a portion of the hydrate), filtering and titrating the filtrate, or measured fraction of it, with standard acid and methyl-orange. The result gives "hydrate + cyanide," from which the hydrate is obtained by deducting the equivalent of cyanide. The precipitate of BaCO₃ is washed thoroughly with boiling water, titrated with standard acid and methyl-orange which gives the equivalent of "carbonate + $\frac{1}{2}$ bicarbonate." By adding an excess of alkali hydrate free from CO₂, before addition of BaCl₂, the whole of the carbonate and bicarbonate may be precipitated as BaCO₃. If the solution be now filtered and tested as before the difference between the amount of hydrate added and the amount found after precipitation with BaCl₂ gives the equivalent of the bicarbonate, and titration of the washed precipitate gives the equivalent of "carbonate + bicarbonate" (for further details see Sutton, 8th edition, pp. 60-63).

Base Metals

Zinc. — Zinc may be determined by ordinary methods of analysis after decomposition of the cyanogen compounds. Evaporate 100 c.c. with 5 c.c. conc. HNO₃ and 5 c.c. conc. H₂SO₄ until fumes of SO₂ are given off. Add 2 to 3 c.c. strong HCl and dilute to 15 to 30 c.c. Boil till all soluble matter is dissolved, add excess of ammonia and boil again. Filter off any precipitate of ferric hydrate, silica, etc. If copper is present acidulate filtrate with HCl in slight excess, and add some strips of lead; or acidulate with H₂SO₄ and add sheet aluminum, boil until solution becomes colorless, filter, and determine zinc in the filtrate by any ordinary method. If iron is present after adding HCl, neutralize with sodium hydrate, and again acidulate very carefully and slightly with HCl. Boil, add sodium acetate in slight excess, boil; this precipitates iron and aluminum as basic acetates. Wash precipitate thoroughly several times by decantation with hot H₂O before transferring to filter paper. For accurate determination, precipitate should be

INGREDIENTS OF CYANIDE SOLUTIONS 247

dissolved and re-precipitated. The filtrate should be slightly acid, clear and colorless. Of the gravimetric methods of estimating zinc, precipitation as carbonate and weighing as oxide is best. Other methods of estimating zinc in cyanide solution are: precipitation as ferrocyanide in acid solution; direct precipitation as sulphide on direct addition of an alkaline sulphide to cyanide solution; and others.

Copper. — To 100 c.c. of the solution add 5 c.c. of conc. HNO_3 and 5 c.c. conc. H_2SO_4 and boil till white fumes of SO_2 are freely given off. Then add ammonia in slight excess, boil, and filter. If much iron is present, redissolve precipitate in HCl and precipitate with ammonia, filter, add second filtrate to first. Determine copper by any method for copper.

Calcium. — Acidify 50 to 100 c.c. of solution with 10 c.c. HCl , boil, filter, make filtrate alkaline with NH_4OH , boil, filter. Mix with a boiling solution of ammonium oxalate, allow to stand till clear (3 to 1 hour); filter and wash precipitate with hot water and titrate with $KMnO_4$. For this purpose, precipitate, after washing free from oxalates, is returned to flask in which it was originally precipitated, moisten paper and funnel with 10 c.c. of 25% HCl and allow washings to run into flask, heat liquid to boiling, dilute to 50 c.c. with distilled water, add 5 c.c. conc. H_2SO_4 , heat to 70° C. and titrate with N/20 $KMnO_4$ (1.5803 gr. per liter) of which 1 c.c. = 0.001 gr. Ca = 0.001% Ca on 100 c.c.

Reducing Agents

Manganese. — Acidify 100 c.c. of solution, pretty strongly, with 5 to 10 c.c. HNO_3 , boil, add peroxide of lead and boil, make up to definite volume in a measuring flask, by adding boiled distilled water. When clear, draw off aliquot part, compare tint with that of a similar volume of water to which standard permanganate is added until colors are alike. The Mn content of permanganate being known, that in solution tested is easily calculated. A convenient standard solution contains 0.1435 gr. $KMnO_4$ per liter. 1 c.c. = 0.00005 gr. Mn.

Reducing Power. — A rough idea of the relative quantities of reducing agent in various solutions may be obtained by comparing the amounts of standard $KMnO_4$ required to give permanent tint to equal volumes of solutions tested, under the same conditions, and acidified in each case with H_2SO_4 . When the finishing point is indefinite, add to acidulated solution an excess of $KMnO_4$, allow to stand for some time, add an excess of KI to the pink liquid until the color changes to brownish-yellow; iodine is liberated in proportion of excess of $KMnO_4$ present. The iodine is then titrated by means of standard thiosulphate, using starch indicator. From this, reducing power is calculated.

Ferrocyanides. — With no other reducing agent present, ferrocyanides may be fairly accurately determined by converting to ferricyanide in an acid solution by $KMnO_4$. If much ferrocyanide is present, add about 100 c.c. H_2O for every 0.1 gr. of ferrocyanide present. A N/10 solution of $K_4Fe(CN)_6$ contains 42.2358 gr. $K_4Fe(CN)_6 \cdot 3H_2O$ per liter, corresponding volume for volume with N/10 permanganate containing 3.1806 gr. $KMnO_4$ per liter. In practice a N/20 or N/100 solution is more convenient. The finishing point is shown by the change from yellow to reddish-yellow. Other methods are available. In the presence of zinc — heat solution to 80° C., make strongly alkaline with $NaOH$, add Na_2S in slight excess, agitate, settle, filter and wash well with hot water. The filtrate is practically free from zinc. Cool the solution, acidulate with H_2SO_4 , and add N/10 $KMnO_4$ till FeS and other reducing agents are completely oxidized, and a faint reddish tint is permanent. Titrate with standard copper sulphate solution using ferrous ammonium sulphate as indicator, in spots, on a white plate. The end point is sharp and definite. Accurate Method: evaporate with acid and run for iron.

Thiocyanates (Sulphocyanides). — Oxidize up with $KMnO_4$ according to — $6KMnO_4 + 12H_2SO_4 + 5KCNS = 11KHSO_4 + 6MnSO_4 + 5HCN + 4H_2O$; 1 c.c. N/100 $KMnO_4$ = 0.00016193 $KCNS$ = 0.00368831

USEFUL DATA

gr. K_4FeCN_6 . It is preferable to use N/10 solution of $KMnO_4$ in this case. When ferrocyanides are present they may be separated by precipitation as Prussian blue with a ferric salt, thoroughly washing precipitate. Thiocyanates may be determined by oxidation with iodine, $KCNS + KHCO_3 + 8I + 3H_2O = KHSO_4 + 6HI + CO_2 + KI + ICN$. A known quantity of thiocyanate is allowed to stand with excess N/10 iodine and 1 gr. $NaHCO_3$ in a stoppered bottle for half hour in dark, and excess iodine titrated with sodium thiosulphate. Colorimetric method: 10 to 20 c.c. solution to be tested, acidulated with HCl , ferric chloride added and the color compared with that produced by standard thiocyanate under same condition. When ferrocyanides are present precipitate formed must be filtered off before tint can be compared.

Sulphides. — To solution add excess of carbonate of lead, oxide of lead, or tartrate of lead in excess of caustic alkali, collect precipitate in filter and wash. Oxidize precipitate by means of chlorine or bromine in presence of an excess of alkaline hydrate, filter and acidify solution. Boil off excess, Cl or Br, precipitate the sulphuric acid by $BaCl_2$ and determine in usual way. Method is accurate but tedious. Another method is to agitate solution with Pb carbonate and filter. The precipitate of lead carbonate and sulphide is transferred to flask and covered with a few c.c. of solution of potassic or sodic cyanide (made from pure potassic or sodic hydrate and pure hydrocyanic acid). Add slight excess of H_2O_2 and allow to act for 10 to 15 minutes. A small quantity of MnO_2 is added, and the mixture agitated for about 2 minutes to destroy excess of H_2O_2 . The solution is then filtered off, acidified with H_2SO_4 , and titrated with N/100 permanganate. They are also determined by adding double cyanide of silver and titrating the free cyanide liberated. The silver double cyanide is prepared by adding silver nitrate to a solution of, say, 0.5 per cent KCN until a slight permanent turbidity is formed, allowing to stand for some time and filtering.

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